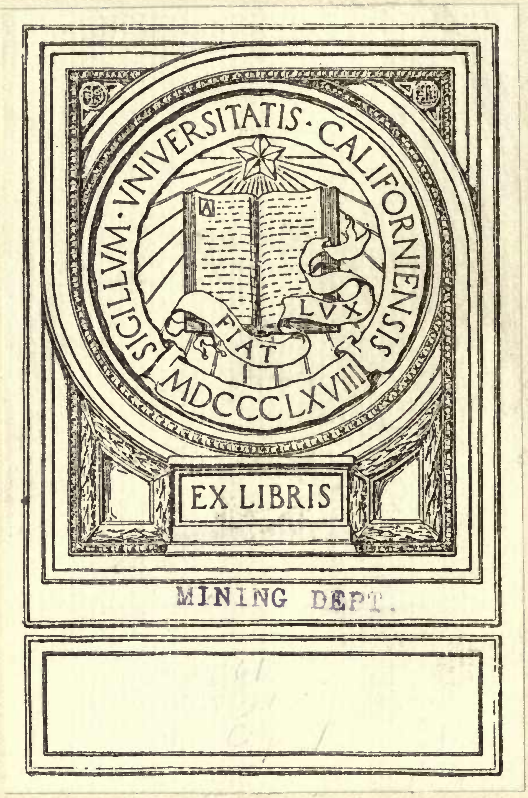
TESTING FOR METALLURGICAL PROCESSES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Testing for Metallurgical Processes

BY

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PREFACE

This book is based upon original notes and experiments made in the laboratories of the Michigan College of Mines, and the nucleus of the material was published in 1909 in pamphlet form under the titles 'Metallurgical Laboratory Experiments' and 'Ore Dressing Laboratory Experiments.' These booklets have been used as text-books in the College. In preparing the present edition the text has been extensively revised and enlarged. Free use has been made of standard reference books, the more important of which are listed below. I am under especial obligations to S. S. Bruce, professor of metallurgy and ore dressing at the Michigan College of Mines, for advice and assistance in preparing the new text, particularly that part relating to slags and slag calculations.

James A. Barr.

Mt. Pleasant, Tennessee,
August 26, 1910.

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AMALGAMATION

Amalgamation is based on the affinity of mercury for gold in the metallic state and consists, briefly, in crushing the ore to the necessary degree of fineness to release the metallic particles and bringing them in contact with mercury with which they become alloyed, or amalgamated. For gold ores in which the precious metal is in a condition to amalgamate with mercury, that is, 'free milling' ores, the amalgamation process offers advantages over all others in simplicity and economy.

In considering the treatment of gold-bearing ore, it is usually necessary to obtain sufficient data to fully answer the following questions: (1) What is the value of the ore? (2) Is the ore free milling? If so, what percentage of the gold can be amalgamated? (3) What value has the tailing after amalgamation? Can it be further treated at a profit? (4) What percentage of concentrate does the original ore carry? Can the ore be profitably concentrated or could the tailing from the amalgamating plates be better treated by other methods?

These questions can be best answered by making a run of 10 to 20 tons of the ore in a regular mill fully equipped for amalgamation and concentration. Before making a trial on such a large and necessarily expensive scale, the following preliminary tests should be made.

Test.—Take a 500-gram sample of the ore and crush gradually so as to pass a 40-mesh screen. Especial care should be observed in regard to any residue or metallic particles left on the sieve. If any such particles are found they should be carefully saved, separately assayed, and the assay value of the final sample corrected accordingly. Carefully mix the 40-mesh ore, take a 50-gram sample for assaying, crush through 120 mesh and determine the gold, silver, base-metal percentages, and amount insoluble.

Free Milling Test.—Take a 5 A. T. sample of the 40-mesh ore, introduce carefully into a wide-mouthed bottle, add clean mercury equivalent to 10% of the weight of the ore taken, with sufficient water to make the whole into a thick pulp. Cork up the bottle securely and rotate from 2 to 3 hours. Separate the mercury from the tailing by a hydraulic classifier, Fig. 1, or by panning with an ordinary gold pan. Save all the water, concentrate, tailing, and slime, allow to stand until the supernatant water becomes clear, then decant the water using care not to lose any of the slime. Dry the remaining tailing, mix well, sample and assay for gold. The difference in the assay of the original ore and the amount found above will equal the gold amalgamated. Calculate the percentage of gold recovered. The gold may be recovered from the mercury as given under 'Retorting.'

If any minerals suitable for saving by concentration are present in the ore, the following tests should be made. Take a 100-gram sample of the tailing from the amalgamation test and carefully pan down so as to save all the heavy minerals such as pyrite. A larger sample may be taken for this test if the result of the concentration does not give a sample large enough for assaying purposes. For this concentration, the ordinary miner's gold pan is used. (Fig. 2.) The weighed sample of ore should be

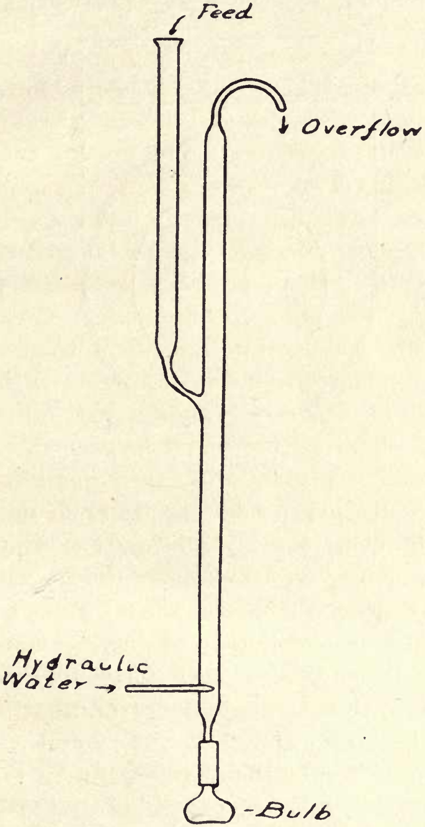

Fig. 1. HYDRAULIC CLASSIFIER

placed in the pan and thoroughly softened with water; the pan is then shaken sideways and in a circular manner to give heavy particles an opportunity to settle; water is flowed across the top, removing the overlying layer of waste. The shaking and flowing are repeated until the concentrate is sufficiently cleaned from gangue. Do not try to pan off the last traces of the gangue, as some of the concentrate will pass over into the tailing. Pan the tailing again to save any concentrate that may have been lost in the first operation. Dry and weigh both the concentrate and the tailing. Assay the tailing for gold and silver. If the value of the gold and silver in the tailing is sufficient, their further treatment by cyanidation should be considered. Assay the concentrate for gold, silver, base metals, and insoluble. Calculate the percentage of concentrate in

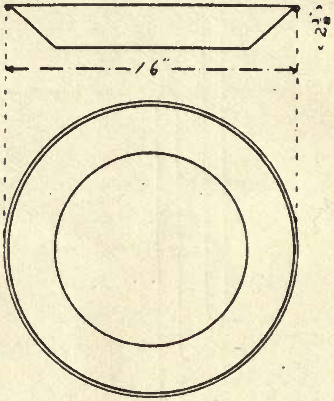

Fig. 2. MINER'S GOLD PAN

the ore; also how many tons of ore will be required to produce one ton of concentrate.*

Make out the following report which will serve as a check on the work:

150 grams of ore amalgamated; gold content as per assay = 0.0250 grams.

	Grams.	Per cent.
Gold recovered, calculated by difference.	0.020	80.0
Gold recovered from 150 grams of ore by concentration	0.0040	16.0
Gold in tailing........................	0.0005	2.0
Gold unaccounted for (by difference)...	0.0005	2.0
Total	0.0250	100.0

Suppose the charge for amalgamation alone is \$0.50 per ton, for concentration and amalgamation together is \$1 per ton, and for concentration is \$0.75 per ton. Allow \$2.50 for mining and \$1 for freight charges to the smelter. With these costs, the data previously obtained, and a smelter schedule, figure the profit and loss in treating the ore:

1. By shipping direct to smelter.
2. By amalgamation and wasting tailing.†
3. By amalgamation, concentration, and shipping concentrate to smelter.

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*Percentage concentrate $= \frac{\text{wt. concentrate}}{\text{wt. ore taken}} \times 100$. If an ore contains 4% of concentrate, 25 ton of ore will concentrate into 1, for 100:4::×:1.

†If sufficient gold was not recovered by amalgamation to pay the cost of amalgamating after crushing (\$0.20), that process would probably be omitted in the treatment of the ore.

Include all information and useful data obtained, in a concise report with remarks as to the best probable method of treating the ore in question.

CHLORINATION

GENERAL

The chlorination process, which is based upon the solubility of gold in chlorine water, offers an available means for extracting the gold from some classes of ores that are not amenable to copper-plate or pan amalgamation, or to the economical treatment by cyanidation, lixiviation, or smelting. With some classes of ores, the chlorination process may give better results than treatment with cyanide. As a rule, the cyanidation process is better adapted for the treatment of low-grade ores, while for the treatment of many higher-grade refractory ores, chlorination gives better results. The treatment of such material by chlorination involves the roasting of the ore which is usually the most expensive part of the operation.

Ores, in which the gangue is hydrated oxide of iron, are extremely difficult to amalgamate, either by plate or pan amalgamation, as the gold is not only very finely divided, but the pulp is usually excessively slimy, enveloping the particles of gold and forming an almost impervious coating on the amalgamation plates. For this class of ores, barrel chlorination gives very satisfactory results. For extracting gold from some classes of concentrate, especially sulphides, and also for the treatment of arsenical pyrites, the chlorination process has been successfully applied. Some classes of pyritic ores contain an amount of acid making the economical treatment by the cyanide process prohibitive, but the same ores yield excellent results with chlorination. The chlorination process can be advantageously utilized for treating ores containing particles of so-called 'rusty gold' or particles coated with other substances that make them refractory to amalgamation, and which at the same time are too coarse for cyanidation. Ores containing much lime or magnesia can not be successfully treated by the chlorination process as there would be an excessive consumption of chlorine by these constituents.

THE OXIDIZING ROAST

Since roasting is the primary step in the process, it will be described in detail first. The object of roasting is to burn the sulphur of the ore to sulphur dioxide, SO_2, which passes away as a gas, and to change into an oxide the metal with which this sulphur was combined. It is not possible in practical work to drive off all the sulphur or to change all the metal into oxide. Moreover it takes a disproportionate amount of time and heat to remove the last per cent of sulphur. In the case of roasting an ore preparatory to the smelting process, it is not desirable to oxidize all the sulphur present in the ore, but in roasting an ore preparatory to chlorination or cyanidation, it is sought to remove most or all of the sulphur, to have the least amount of sulphates, and as much of the metal in the oxide form as possible.

The requirements for good roasting are:

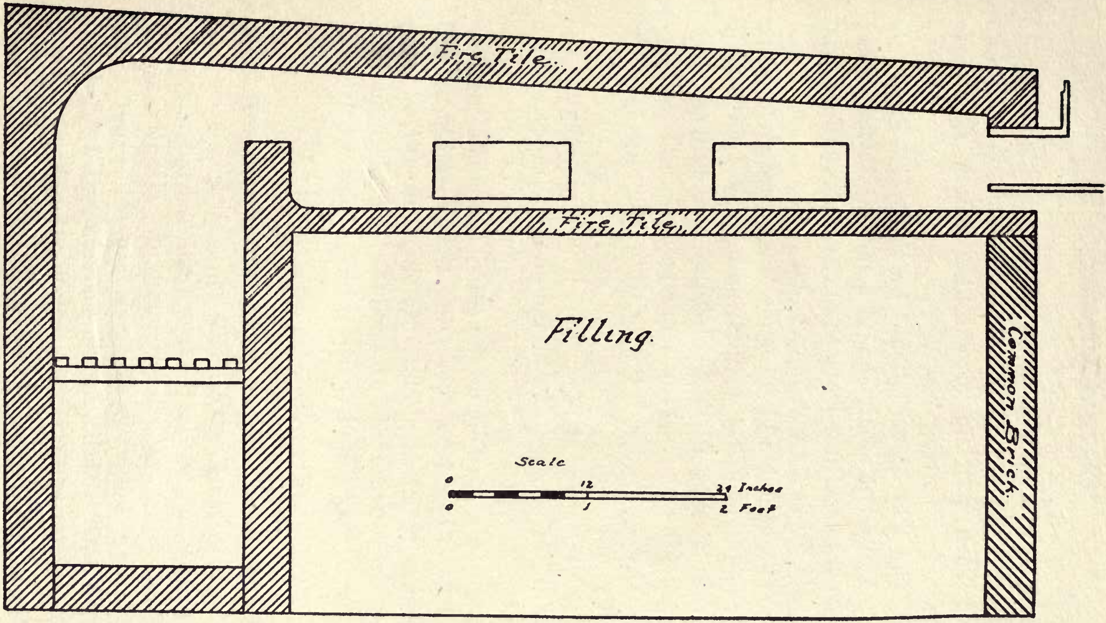

Fig. 3. LABORATORY ROASTING FURNACE

1. The ore should be in a finely divided form so that the oxygen of the air can come into contact with each particle of sulphide which should be stirred frequently.

2. The bed of ore should be thin.

3. A current of air should constantly sweep away the gases resulting from the roasting reactions, and bring a fresh supply of oxygen.

4. The ore should be sufficiently heated to bring about the reactions desired, and yet not enough to melt the easily fusible sulphides. A dull red heat answers the above requirements.

5. The gases used for heating the ore must be highly oxidizing. Since the surface of the ore is experiencing rapid oxidation, it is plain, that if the expanse of surface can be increased without enlarging the hearth area, a larger quantity of ore can be roasted in a furnace of the same size. The increase may be effected by furrowing the ore with the aid of rakes or small ploughs, so that, instead of lying in a smooth horizontal bed, the ore shall be in a series of ridges and furrows like a plowed field.

SOLUTION OF THE GOLD

The solution of the gold is effected either in tanks or in revolving barrels.

In the former method, which is known as the 'Plattner process', the roasted ore, slightly moistened with water, is charged into acid-proof tanks supplied with tightly fitting covers and a false bottom or filter-bed. Upon receiving the charge of ore, the covers are replaced and hermetically sealed, after which chlorine gas is forced into the moist ore, and converts the gold into the soluble tri-chloride according to the reaction:

$$Au + 3Cl = AuCl_3.$$

The gold chloride formed is leached out with water. The gold-bearing solution is run into settling tanks where the gold is precipitated and collected.

In the second method, known as the 'barrel process', the moist ore is charged into lead-lined barrels fitted with a filter-bottom. The chlorine is generated directly in the barrel by sulphuric acid and bleaching powder according to the reaction:

$$H_2SO_4 + CaOCl_2 = CaSO_4 + H_2O + 2Cl.$$

During the solution process, the barrel is continually rotated. The auric chloride is leached out with water as in the previously described process.

PRECIPITATION

The precipitation may be effected either by the addition of ferrous sulphate or hydrogen sulphide. The latter method is more generally used. The excess of chlorine gas is first eliminated by sulphur dioxide:

$$2Cl + SO_2 + 2H_2O = H_2SO_4 + 2HCl.$$

The gold is precipitated by hydrogen sulphide.

$$2AuCl_3 + 3H_2S = Au_2S_3 + 6HCl.$$

The resulting precipitate is collected in a filter-press, dried and refined.

PROCEDURE FOR THE TEST

Take a 500-gram sample of ore and assay for gold, silver, base metals, and sulphur.

The ore should then be crushed so as to pass through a 40-mesh screen, carefully weighed and placed in an iron roasting dish in a thin, even layer. The success of this test depends upon the roast being 'dead'; that is, the ore should be roasted until practically all the sulphur is driven off.

The roasting dish, containing the ore, should then be placed in the oven or reverberatory furnace. The temperature of the furnace should be gradually increased until the sulphur is seen to take fire and burn with a blue flickering flame, at which point the temperature is held stationary until the sulphur ceases to burn of its own accord. Rabble the charge frequently so as to prevent caking. Carry on the roast with continually increasing temperature up to the maximum limit allowed by the nature of the ore, until the sulphur content does not exceed 0.2 or 0.3% (to be determined by analysis). This roast will take from 4 to 5 hours. During the roast, endeavor to keep the mechanical losses of the ore as low as possible.

The decomposition of copper and iron sulphates is complete at 850° C., but this is by no means an instantaneous action. Any ferrous sulphate left in the roast would be oxidized to the higher form, which action is not harmful except to consume chlorine. The reaction is,

$$6FeSO_4 + 3Cl_2 = 2Fe_2(SO_4)_3 + Fe_2Cl_6.$$

Undecomposed sulphides and charcoal are detrimental, as they precipitate the gold prematurely according to the following reactions:

$$2AuCl_3 + 3CuS = Au_2S_3 + 3CuCl_2, \text{ or}$$
$$3CuS + 8AuCl_3 + 12H_2O = Au + 24HCl + 3CuSO_4.$$
$$4AuCl_3 + 3C + 6H_2O = 4Au + 12HCl + 3CO_2.$$

After the roast has been completed and the ore cooled, weigh carefully and determine the loss in weight due to the conversion of the sulphides into oxides, in addition to the mechanical losses. The latter will be small enough to neglect if the experiment has been carefully performed. Break up any lumps in the roast and pass through a 30-mesh screen. Sample and assay for gold and sulphur. Calculate the total amount of gold and sulphur in the roasted sample. Having already determined the total amounts in the original sample taken, the losses of each are found by difference, from which the percentages of loss are determined.*

In making the chlorination test introduce 3 A. T. of the roast ore into a strong bottle together with the following charge: bleaching power (35% Cl), 6%†; sulphuric acid, commercial, 12%; water as required.

The quantity of bleach required varies with the nature of the ore and the economical amount can only be found by trial. The quantity of acid required for a given amount of bleach is variable. Theoretically, 7 parts of bleaching powder require 6 parts of acid, but in practice this proportion is seldom used, the proper amount being determined

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$$*\frac{\text{Loss S in roast}}{\text{Wt. in original}} \times 100 = \text{per cent loss of sulphur.}$$

†Six per cent of the weight of ore taken.

by the character of the ore. An amount of acid in excess of the theoretical requirement is recommended, otherwise sulphate of lime is liable to be precipitated with the gold during subsequent operations.

A method of charging barrels in practice is to add first the requisite amount of water, then the sulphuric acid, after which the barrel is charged with the roasted and pulverized ore, and lastly the bleaching powder. By this method the chlorine is not generated until the barrel is sealed and rotated. In the laboratory, to the roasted ore add enough water to make the whole charge into a thick pulp. The bleach, wrapped in a piece of filter paper, is added, the cork put tightly in place and securely tied down. Wrap the bottle in some burlap to prevent trouble in case the bottle bursts. Rotate the bottle containing the charge from 4 to 6 hours, at the end of which time it may be again opened, when the odor of chlorine should be very noticeable if sufficient chemicals have been added to the charge.

Leach the charge by transferring to a large filter paper held in a ribbed funnel and adding wash-water until all the gold chloride has been washed out. The gold may be recovered from the solution by precipitating with hydrogen sulphide, filtering off the resulting precipitate, scorifying with test lead and cupelling for gold. Dry the leached ore, sample and assay for gold. The amount of gold extracted is calculated from the difference in the assays of the roast ore and the leached product.

Make out a report in the following manner:

	Weight	Silver	Gold	Sulphur	Remarks
Original ore.....					
Roasted ore......					
Losses in roast..					
Leached ore.....					
Extraction					
Actual recovery..					

Prepare an itemized statement of costs, including profit and loss. Show profit by shipping ore or concentrate direct to smelter.

CYANIDATION

In making tests on gold and silver ores to determine if they are amenable to treatment by the cyanide process, the greatest care, accuracy of manipulation, and good judgment are necessary. A correct and representative sample is the first and most important consideration. Among other points to be carefully investigated are the physical and chemical states in which the gold and silver exist; the proper methods of grinding the ore; the fineness of grinding; the presence of minerals suitable for concentration; the solubility of gold and silver in the cyanide solution; the consumption of chemicals; the precipitation of the dissolved metals. Throughout all the investigations the question of profit and loss must be kept constantly in view. As a rule the experimental work on a small scale, such as in bottle tests, should not be taken as conclusive nor the data used to construct a plant for the commercial treatment of the ores. They should rather serve as an aid to larger tests on the ore in lots of 5 tons or more. Generally speaking, the percentage of extraction and the consumption of cyanide will be larger in the laboratory tests than in a regular mill-run.

PRELIMINARY INVESTIGATIONS

Take a sample of 30 to 40 pounds of the ore, from which take a smaller sample of 2 pounds for the following tests.

1. Make check assays for gold and silver and thus

establish the value of the ore. If below $5 per ton gold the conditions must be very favorable to permit cyaniding the ore.

2. Examine the ore under a low-power microscope and determine as far as practicable the physical state of the gold and silver and the associated minerals. If the gold appears to have a smooth even surface, or appears in well-defined crystals, it may be taken that the cyanide will, in all probability, dissolve only a small portion, and amalgamation will be necessary if these particles are to be extracted. If, on the other hand, the gold is spongy and honeycombed in appearance, finely divided, or invisible, it is possible that it may be treated direct by cyanide. Then, again, the matrix may be of such a nature that crushing produces an excessive quantity of slime, as, for instance, when an ore is clayey or consists of certain shistose rocks, in which case wet crushing might produce, for the most part, a product impervious to the solution, while with dry crushing leaching might become practicable with a larger percentage of ore. The ore might be mainly quartz and then either wet or dry crushing could be adopted. The decision would be influenced by the local conditions and the physical state in which the gold exists. When silver is present in chemical combination the microscope will often show which compound is acted on by the solution and which is not. This information may direct in an important manner the treatment method.

The presence of metallic mineral matter, such as sulphides of iron, copper, antimony, or zinc, arsenides and tellurides, should always be determined by microscope. Their systems of crystallization and the particular minerals present should be identified, as far as practicable, in every unknown ore. This examination may lead to important results bearing on the method of treatment and save much time in experimenting. In cases of failure such an examination may indicate the cause and methods of overcoming the difficulty. The microscope will assist in determining whether the coarse particles of gold are amalgamable before or after cyanidation.

CHEMICAL TESTS

1. CONSUMPTION OF CYANIDE

Introduce 20 grams of ore, treated with a sufficient quantity of alkali, if necessary, as found in (1), into a glass-stoppered bottle; add 40 c.c. of KCN solution of known strength, agitate for 20 minutes; filter; measure off 20 c.c. of the filtrate and titrate for undecomposed KCN. The difference in the amount of KCN in 20 c.c. of the stock solution and the amount found above, gives the amount of KCN consumed by 10 grams of ore. Compute in terms of pounds of KCN per ton of ore.* If not over 4 pounds of KCN per ton, for an ordinary grade of ore, the succeeding tests may be tried.

Tests (1) and (2) may often be combined to advantage as follows: Weigh out 4 separate 20-gram samples of pulp, place in glass-stoppered bottles; add

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*Method of calculation.—Suppose 10 grams of ore consumes 0.01 grams of KCN; which is 0.1% of the weight of ore, 0.1% × 2000 = 2 lb. KCN per ton of ore.

fresh-slaked, pure lime, at the rate of 5, 10, 15, and 20 pounds to the ton of ore, and then 40 c.c. of KCN solution of known strength, to each bottle; place on the agitator for 20 minutes, filter, and determine the cyanide consumption as given above. If a high consumption of cyanide is shown even with 15 to 20 pounds of lime per ton, see if the soluble salts can be removed by a preliminary water wash, using 3 washes, each double the volume of the ore.

2. TOTAL ACIDITY

Gold and silver ores usually contain acid products as a result of oxidation of sulphides and arsenides, that are soluble in water and have the power of neutralizing free alkali. In some cases it may be profitable to give the ore a preliminary wash with water to get rid of most of the acid. Ordinarily the acid is neutralized by adding an alkali.

Test.—Agitate 20 grams of the pulp for 20 minutes in a well-stoppered bottle, with 40 c.c. of tenth normal sodium hydroxide solution, which should be an excess of alkali; filter off one-half of the solution added, and titrate the free alkali with tenth normal acid. Report the amount of lime necessary to neutralize a ton of ore.* The consumption found by the above titration will be for 10 grams of ore.

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*Method of calculation.—10 c.c. of acid required for titration; 20 c.c. NaOH contains (a) grams NaOH; 10 c.c. 10/N acid = 10 c.c. 10/N alkali; 10 c.c. 10/N NaOH contains (b) grams NaOH; a—b = grams NaOH consumed by 10 grams of ore.

Then 10 (a—b) = NaOH consumed by 100 gram of ore or 10 (a—b)% × 2000 = consumption NaOH per ton of ore, which is converted into terms of CaO.

3. BEST STRENGTH OF SOLUTION

Take 3 four-ounce, wide-mouthed glass bottles, place 1 A. T. of pulp in each; then add 60 c.c. of KCN solution of the following strength, respectively:

To No. 1 add 0.05% KCN
“ “ 2 “ 0.25 “ “
“ “ 3 “ 0.75 “ “

Before adding the KCN solution, thoroughly mix the proper amount of neutralizer as found in (1). Allow to stand for 48 hours with occasional shakings. Filter off equal portions of the solution and titrate for undecomposed KCN, from which compute the consumption of KCN per ton of ore. Thoroughly wash, dry, and assay the tailing for gold and silver. From the assay of the original pulp and the assays of the tailing, compute the percentages of extraction in each case.

Physical Effect of Strong and Weak Solutions.—In practical work, the strength of solution was found, in some cases, to have a marked physical effect on the dissolving of gold and silver. This is most noticeable in ores containing a preponderance of silver over gold in the presence of sulphides. Thus it was found when employing a weak solution of 0.15% strength KCN a maximum extraction of 66.6% silver was reached, and even when the solution was afterward strengthened to 0.50% the additional extraction was very small. When, however, the first solution applied was 0.50% strength and afterward weaker solutions were used, the extraction was over 90%. It is surmised that the weak solution produced a hard, insoluble film of sulphide over the surface of a portion of the metallic particles which prevented even the strong solution from attacking the silver, whereas, when the strong solution was used at first, the sulphide adhered as a loose slimy deposit, which did not prevent the weaker solution from afterward attacking the metallic silver. A hot solution produces the same slimy deposit even when weak.

Comparative Solubility of Metals and Minerals in Cyanide Solutions.—It has been found by experiment that the rate of dissolution of gold and silver varies with the strength of solution, being small for strong solutions and increasing as the solution becomes weaker until a maximum at 0.25% strength of KCN is reached, and then again diminishing. These experiments were performed by Maclaurin with plates of pure gold and silver, which accounts for the metals not being as soluble as in ores. This is accounted for by the unavoidable polarization of the gold and silver plates by hydrogen bubbles that are formed during the experiment. The curve, Fig. 4, clearly shows the comparative solubility of the different metals and minerals in cyanide solutions of different strengths and the efficiency of KCN per unit in each case. On studying the table (Fig. 4) it will be seen that no proportionality exists between the weight of metal dissolved and the percentage strength of solution. Thus a 0.01% solution dissolves 24 parts of gold, whereas a 0.1% solution, ten times as strong, dissolves only 82.5 parts in that time. Therefore, the weaker the solution the greater is the efficiency per molecule of KCN dissolved, but

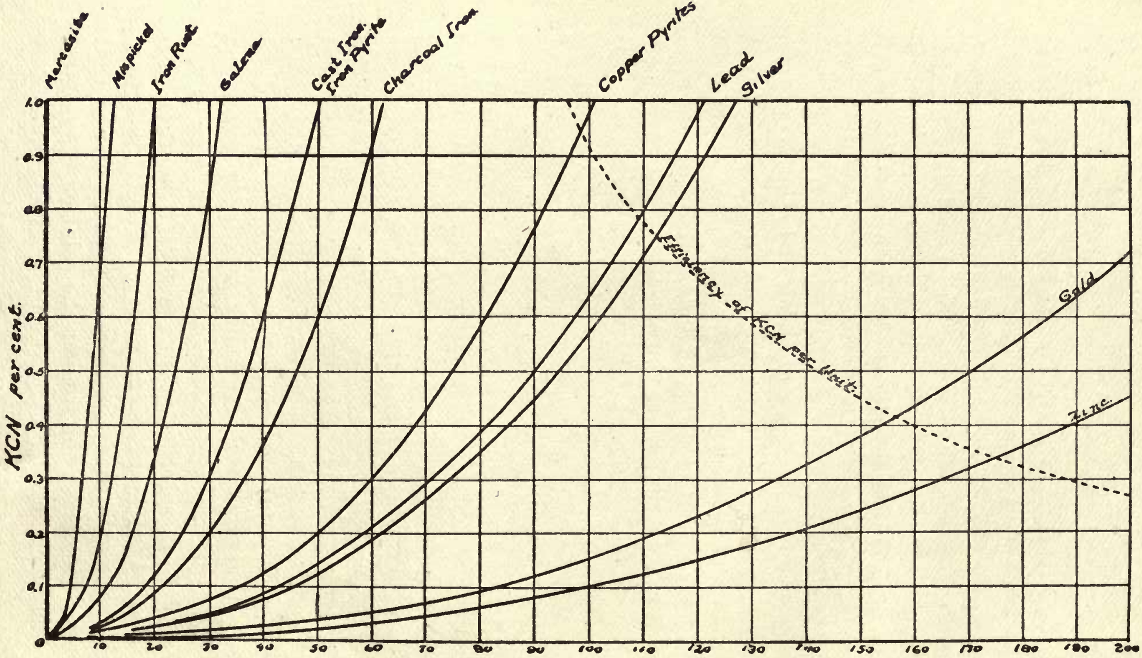

Fig. 4. UNIT BY WEIGHT DISSOLVED IN EQUAL SPA CES OF TIME

in practical work this rule can easily be carried too far. A solution that would have the maximum dissolving power and a maximum efficiency per molecule of KCN is desired for working strength. This can, however, be arrived at only in an empirical way as the time factor has to be taken into account.*

TABLE 1.

Order of Solubilities	Per Cent KCN									
	0.1	0.09	0.08	0.07	0.06	0.05	0.04	0.03	0.02	0.01
Gold	82.5	78.5	74.0	60.25	64.0	58.0	51.5	44.5	35.5	24.0
Silver	45.5	43.3	40.8	38.2	35.3	32.0	28.4	24.6	19.5	13.2
Efficiency of KCN per unit	0.344	0.363	0.385	0.412	0.444	0.483	0.536	0.618	0.740	1.000

Now if the time required to treat an ore with solutions of known strength be known, and it be desired to vary the capacity of the works by lengthening or shortening the time of treatment, what strength of solution would give equally good results may then be determined from the data given. Thus if it required 200 hours to treat an ore effectively with a 0.01% solution, and it became necessary to increase the capacity of the works by shortening the time of treatment to 70 hours without decreasing the extraction, it would be necessary to find in the column a dissolving efficiency unit which multiplied by 200 equals 70 or thereabouts. Then the strength of solu-

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*If the time factor is left out of the question, as in cases where the gold and silver dissolve rapidly but can be washed out of the ore only slowly, the most efficient strength of solution is then from 0.07 to 0.09% KCN.

tion at the head of the column is the one required. In this case it will be found that $0.344 \times 200 = 68.8$. The required strength found at the head of the column is 0.1 per cent.

4. FINENESS OF GRINDING

Take 3 samples of the ore ground so as to pass through 20, 40, and 100 mesh, respectively; place in bottles with the required amount of alkali and cyanide solution as determined in the previous tests; agitate for 48 hours; filter; wash; dry; assay the tailing. Calculate percentage of extraction for each mesh.

The object is to break the ore into particles of such size as will leave the gold and silver capable of coming into contact with the cyanide solution. This end is obtained by fine pulverizing, and much may be learned by examining the grains under a low-power microscope to determine the minimum degree of fineness necessary; a point which, however, must ultimately be decided from empirical results by testing with cyanide solution. The more finely the ore is crushed, the more perfectly does the gold and silver content dissolve, but if crushed too fine an excess of dust or slime forms, which interferes with the separation of the dissolved gold and silver and causes an increased consumption of cyanide. If crushed too coarse a large percentage of gold and silver remains locked up and is not dissolved, but that portion which does dissolve may be separated more or less readily. Under no circumstances can an ore be crushed to one uniform size, but the aim is to crush it so as to get a maximum number of those particles that give the highest extraction at the lowest cost. This can be found only by experiments. In many cases it is advantageous to crush wet, while in others dry crushing may be more beneficial or profitable.

5. TIME OF TREATMENT

Take 4 samples ground through the mesh as determined in (4); add alkali and cyanide solution as given under (2). Agitate occasionally for 1, 3, 5, and 10 days, respectively, with the free access of air. Determine the percentage of extraction for each sample. Construct a curve as shown in Fig. 5 to show the rate of dissolution of the gold and silver. The curve will help to indicate when the profitable time limit of treatment would be reached.

In an ore the variety of sizes and shapes of the particles of gold must be very great, but a consideration of a few assumed cases will assist in understanding what actually takes place in practical operations. (1) Where the gold is in thin plates the rate of dissolution will be uniform until complete. Where gold is imbedded in pyrites so that only one edge can be attacked, it would also have a constant rate of dissolution. (2) In the other extreme case where the gold is assumed to exist in spheres, then in equal intervals of time, equal thickness of shell will be dissolved but as each successive shell is of smaller diameter the amount of gold dissolved in an equal space of time will be continually decreased. (3) If, however, in order to approximate more closely to actual conditions, a mixture of spheres of various sizes be assumed, it is clear that the smaller spheres

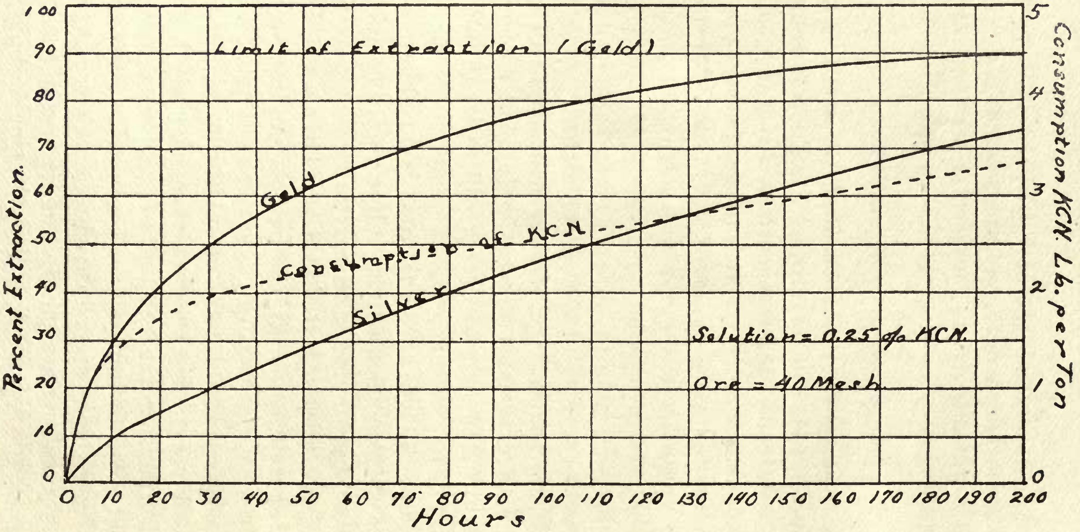

Fig. 5. DIAGRAM OF EXTRACTIONS

will be entirely dissolved before the larger ones and in such cases the rate at the beginning will be considerably increased.

Percolation and Leaching.—The process of causing a liquid to pass through a porous medium such as sand is termed **percolation**, and when the liquid, as it percolates, dissolves some of the porous medium, it is said to **leach.** The advantages of the percolation method of leaching in tanks are, that enormous quantities may be dealt with in one operation, and by the simplest possible plant at the lowest cost; also the method admits of easy and close control.

6. RATE OF PERCOLATION

This varies with the volume and uniformity of the interstices between the ore particles, and to some extent with the depth of the column, the pressure, and temperature. The constituents of the ore also have some influence on the rate of percolation, for some minerals such as galena, pyrite, and blende have a marked effect in diminishing the rate of percolation, and this therefore varies with the proportion of these minerals to the silicious particles present. It naturally follows that the coarser the ore particles are, the larger will be the pores or interstices and the more rapid the rate of percolation. Then, again, uniformity in the size of the ore particles has something to do with the rate of percolation and the leaching effect, for if an ore be crushed so as to pass through a 90-mesh screen, and washed free of slime, a good leachable product may be obtained. If the fine ore be mixed with coarser particles, percolation becomes slower than when the two sizes are treated separately, and leaching is retarded.

The loss in velocity as the solution descends into the ore, seems to be due in a large measure to the pushing forward of air contained between the particles, but also the skin friction between the solution and the ore. In its attempt to escape, some of the air moves in an upward direction and sometimes reaches the surface, but a very large number of the globules are retained, as the pressure does not enable them to force their way upward and escape. These remain in the ore, balanced with the weight of solution above. Such a state of things has a marked detrimental effect on the rate of leaching and causes an excess of moisture to be left in the ore after draining. With upward percolation these globules are not formed to the same extent as in the downward method. The chief quantity of air below the ore surface escapes through the outlet pipe below the filter, and if this outlet pipe is not open a large volume of air bubbles upward, through the sands, affecting the texture of the ore.

The amount of moisture retained by a leached ore depends on the size of the particles, the uniformity of texture, air pressure, and temperature, and to some extent on the depth of the ore column. Temperature has an effect, which in some cases is marked. The higher the temperature, the smaller is the quantity of moisture retained. Bad leaching is chiefly the result of uneven percolation, and uneven percolation follows from want of uniformity in the texture of the ore, or from want of uniformity in the pores of the filter cloth. Whenever the filter cloth becomes clogged with slime the extraction becomes very poor.

There is no fixed rule for applying solution to the ore. Some operators apply the solution in three or four large charges, while others apply it in many small charges. Theoretically the latter method should give the better extraction, but in practice the former gives equally good results. It is in the final treatment, when the water wash is applied, that a number of small charges are highly beneficial. Three or four small charges of 10 tons each will extract more salt absorbed by the ore than one large charge of 30 to 40 tons. This is noticeable in the first small charge of water wash, which after percolating will be found, when drawn off, to be richer in alkali, cyanide, and gold than the previous weak solution. The cause of this is not clear, but it would appear to be due to some reaction between the absorbed salt and the air drawn into the ore mass as the solution drains off, whereby the absorbed salt becomes more soluble.

If the ore is to be leached by percolation the solution must sink through the column of ore with sufficient rapidity to keep the time of treatment of one lot of ore within commercial limits. The rate of percolation is determined sometimes by the time the solution above the ore sinks to a certain depth; thus over 3 inches per hour is good, 1½ inches is fair, and ¾ of an inch is poor. Less than ¾ of an inch is usually uneconomical. A better method is to judge the porosity of the ore by comparing the quantity of moisture retained after draining. The greater the percentage of moisture retained the smaller the porosity and the slower the percolation.

Since the cost of leaching in tanks is less than by any other means, it is therefore advisable to treat the largest quantity practicable by this method. In some classes of ore the quantity of slime is of such a character and so small after reducing the ore to the required fineness that, when mixed evenly with the leaching products, it does not materially interfere with the percolation, or with the extraction of the metal. In such cases it might be most economical not to size the ore before treatment. In case it is found necessary to size the ore to secure the necessary rate of percolation, the maximum quantity amenable to this method of treatment is determined by first sizing a sample and testing each size separately, when the limit will be found at which percolation in tanks must cease.

Test.—To compare the porosity and rate of percolation, the apparatus shown in Fig. 6 may be used. (A) is a tube from 1 to 2 inches in diameter and 4 or 5 ft. long, connected at its lower end to a measuring glass (B) by means of rubber tubing (C). The tube (A) is filled to a mark, say 4 ft. from the bottom, with ore previously dried and the net weight noted. A rubber stopper (a) fitted with a short length of glass tube serves as a means of connection (c). A piece of muslin is placed over the stopper (a) to support the sand and act as filter-bottom. The glass (B) and the tube (c) are filled with clean cyanide solution of work-strength and zero mark is brought to the level (a). The cock (T) is opened and (B) is very gradually raised until the solution in (A) rises to a point about three inches above the ore surface and the solution in (B) is on the same

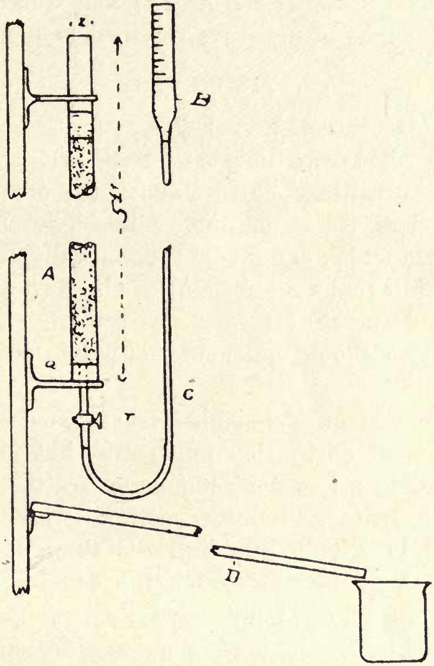

Fig. 6. APPARATUS FOR TESTING THE POROSITY OF LEACHING MATERIAL

level. The amount of solution left in (B) is read, thus giving the volume of solution necessary to cover an amount of ore equal to the volume contained in the tube (A). (B) and (c) are next lowered and the solution allowed to drain off. Time observations are then made at the rate of fall of the solution. Allow the ore to drain for as many minutes per foot of depth as would be employed on a large scale. Finally disconnect (C) from (a) and weigh (A) to find the weight of moisture retained in the ore.

7. SIZING TEST

In case the rate of percolation as found above does not come within the limit set for the ore, it will generally be advantageous to classify the ore into two or more sizes, coarse or fine. Slime may be defined as that ore which cannot be economically treated in the leaching tanks on account of the fineness of the particles. The next larger size resulting from the classification should not hold more than 40% moisture.

The process of wet-sizing is best carried out in some form of elutriation apparatus, but it may be performed in a less perfect manner by the use of a gold pan. In case the latter method is used, the ore is placed in a gold pan and the slime floated off from the sand in very much the same manner as given on page 12. A much more satisfactory result is obtained by employing a small elutriation apparatus, as shown in Fig. 7. The apparatus consists of inverted cones having an angle of about 33° at their apexes, and having diameters of bases in the proportion of 1: 2: 4. Other proportions are not of much importance. The vessel (A) is adjusted to a suitable height and water is allowed to flow into it a little faster than it flows out through the tube (a), and is thus kept overflowing. When the apparatus is filled with water, a clip (s) is closed and a small quantity of ore in the form of sludge is fed from the funnel into the first cone by loosening the spring clip (p). The clip (p) is then closed and (s) is opened and the water allowed to flow from the cistern (A) until it is practically clear as it leaves the discharge pipe (d). At the apex of each cone is a short tube of rubber fitted with a spring clip for the purpose of discharging the ore into separate vessels at intervals. The operation is repeated until a sufficient quantity of each size is collected. The fine and light particles flow into the cistern (D) where they are allowed to settle. By re-treating the settled portion in (D), seven sizes may be obtained; and if each size is submitted to a percolation test, the proportion of non-leachable material present and in what sizes the gold and silver chiefly exist may be readily determined. Other factors may also be ren-

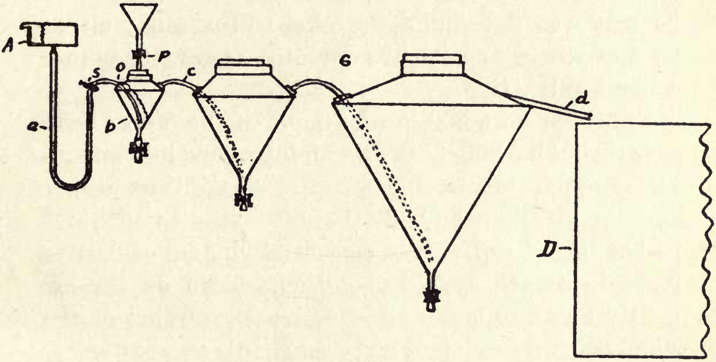

Fig. 7. ELUTRIATION APPARATUS FOR SIZING AND CLASSIFYING

dered apparent, as for instance, whether the ore has been crushed sufficiently fine to liberate the gold and silver. Tests may be made on each size by leaching with cyanide to determine where the bulk of metal remaining in the residues is to be found.

8. SLIMING TEST

In case the solution of the gold and silver is not obtained in the coarser sizes of the ore, tests should be made to determine the effect of sliming all or part of the ore as may be found necessary to obtain a good extraction.

Test.—Grind a 300-gram sample of the ore so as to pass through a 200-mesh screen, introduce into an agitation apparatus, as shown in Fig. 8. Add the proper amount of alkali and fill the apparatus to within 3 inches of the top with a measured volume of 0.01 to 0.05% strength cyanide solution. Turn on the air gently so as to make an effective circulation of the slime but not so violent as to cause loss by spattering. In treating gold ore slime in the laboratory, agitation from 2 to 4 hours is generally sufficient. With silver ores, the time required may be several weeks. During the agitation take a sample by means of a siphon, (one shown in Fig. 8 is very convenient) every hour; assay for gold, silver, and free KCN. Return the unused portion of each sample to the agitation apparatus, also note the amount used for each assay. Keep the strength of the KCN solution up to the standard selected by the addition of known quantities of KCN. From the assays of the samples taken above, determine the percentage of the extraction for each hour and construct a curve from these data as shown in Fig. 5, p. 34. The amount of cyanide

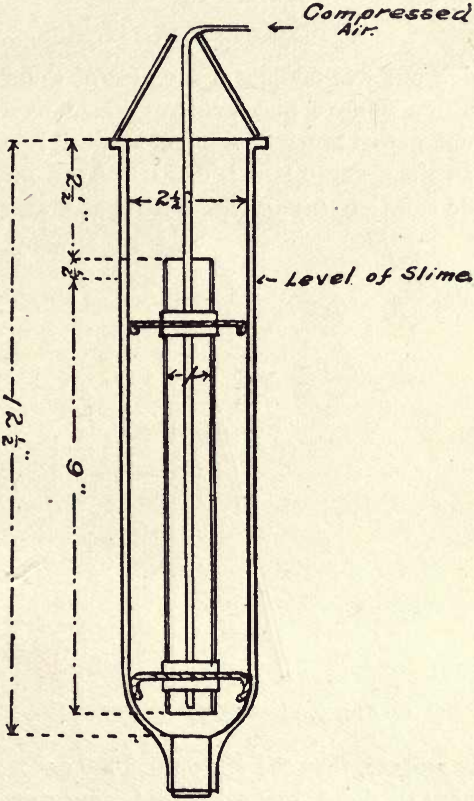

Fig. 8. SLIME AGITATOR

added to keep the solution up to the standard strength will give an approximate idea of the amount of cyanide consumed by the weight of the slime treated. Calculate in terms of KCN per ton of dry slime. See footnote, page 27.

9. DECANTATION TESTS

Weigh out five 500-gram samples of the slime, put each into a 1000 c.c. measuring cylinder together with different percentages of some coagulating substance, such as lime, say 0.1, 0.2, 0.4, 0.6, 0.8%, add dilute cyanide solution, say 0.05%, a little at first, and thoroughly agitate, then fill up to the 1000 c.c. mark. Allow these to stand 3 or 4 hours, taking care that the temperature remain constant, and note the depth of sediment in each case. The best percentage of lime is that which gives the greatest amount of clear solution for decantation. In some cases a maximum

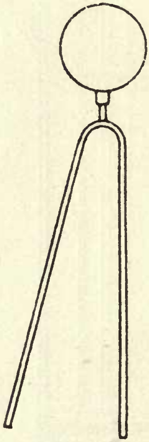

Fig. 9. SAMPLING SIPHON

point is soon reached, while in others the most economical amount of lime has to be determined partly by cost. Beyond a certain amount of lime no advantage is derived, but often subsidence is retarded. Caustic lime is generally employed in the cyanide process, as it protects the cyanide, but it is not so good a coagulating substance as other compounds of calcium. Thus calcium carbonate (chalk) is often more rapid and produces a more dense sediment, which is not so easily disturbed while decanting as when hydrate is used.

10. DETERMINATION OF THE CAUSES OF CYANIDE CONSUMPTION

The cause of a high cyanide consumption may be determined by an analysis of the solution after use. For every part of cyanide rendered inoperative a corresponding part of metal enters solution. Thus one part by weight of iron consumes seven parts by weight of cyanide, and zinc and copper, the number of parts shown in the curve, Fig. 4, page 30. Salts of aluminum and magnesium form hydrates with a liberation of HCN as below:

$$MgSO_4+2KCN+2H_2O=Mg(OH)_2+K_2SO_4+2HCN.$$

This may be overcome by a preliminary alkaline treatment during the crushing, their hydrates being precipitated which are inert toward KCN as follows:

$$MgSO_4+Ca(OH)_2=Mg(OH)_2+CaSO_4.$$

Any free acid has a powerful effect:

$$2KCN+H_2SO_4=2HCN+K_2SO_4.$$

Cocoa matting, jute, and vasculose when new are especially destructive to the solution. Vasculose acts as a reducing agent. The presence of soluble and insoluble carbonates greatly facilitate the decomposition of the cyanides. Metallic iron from the crushing machinery becomes disseminated throughout the ore in fine grains. It dissolves slowly but is nevertheless a factor in the decomposition of cyanide, and the hydrogen thus liberated combines with the valuable oxygen of the solution as follows:

$$Fe + 6KCN + 2H_2O = KFe(CN)_6 + 2KOH + 2H.$$
$$2H + O = H_2O.$$

Pyrite oxidizes rapidly in moist air to ferrous sulphate which is the cause of considerable consumption of cyanide. Most of this injurious substance may be removed by a preliminary water-wash and the remainder neutralized by alkali, forming ferrous hydrate and alkaline sulphate. Ferrous hydrate and the sulphate act as de-oxidizers and consume oxygen that might otherwise be of service in dissolving the gold and silver. Ferrous sulphate oxidizes to normal and basic ferric sulphates, which in time consume cyanide, forming $K_4Fe(CN)_6$ and probably $K_3Fe(CN)_6$.

Copper in the native state and in the form of dynamite caps is also acted upon by cyanide as below:

$$2Cu + 4KCN + 2H_2O = K_2Cu_2(CN)_4 + 2KOH + 2H.$$
$$2H + O = H_2O.$$

A mere chemical analysis is not to be relied upon in testing copper ores, as much depends upon the physical condition in which the copper and gold exist. Sometimes an ore containing from 2 to 3%

of copper may be successfully treated by cyanide, while some ores containing 0.05% and less are failures.

Arsenic is not harmful to the cyanide process, but nearly all of its compounds are soluble in caustic alkali and decompose cyanide in the absence of free alkali. Mispickel (arsenical pyrite, $FeAsFeS_3$) is next to iron pyrite the most abundant metallic compound found associated with gold ores, and is generally the most auriferous of the two. It weathers in moist atmospheres to ferrous sulphate and to hydrate and oxide of arsenic. The latter is only harmful in the absence of free alkali. In treating mispickel ores, it is generally found that the addition of large amounts of lime to the ore effects a large saving of cyanide and improves the extraction. If magnesia is procurable it will be found to answer equally as well, and, as a rule, a better extraction can be relied upon. Sulphides of arsenic, as realgar and orpiment, are both attacked by the alkalies, forming arsenites and thio-arsenites. Alloys of iron and arsenic must be finely ground for a good extraction of the gold and silver. The compounds of alkali and arsenic that form act as reducers, and consume oxygen dissolved in the cyanide solution, and in this way retard the dissolution of gold and silver.

Antimony sulphide, as stibnite, is often found associated with gold ores and is often highly auriferous. Like arsenic, antimony does not form any definite compound with cyanide, but the sulphide is very soluble in caustic alkali and decomposes cyanide. The antimony compounds act as strong de-oxidizers and remove the absorbed oxygen from the solution by forming antimonate and thio-antimonate. Gold and silver may be rendered almost insoluble from this cause; also from the fact that much cyanide is destroyed. Auriferous antimonial concentrate carrying as high as 20% antimony, has been successfully treated by giving the ore an ordinary careful roast, previously mixed with 2 to 5% of charcoal or coal, followed by a hot acid wash of dilute hydrochloric acid (obtained by exposing old chlorine solutions to the direct sunlight) and then lixiviating with cyanide solution. Another method is to roast as above, salt at the end of the roast, and chlorinate. Both methods have yielded 90% extraction but have the drawback of requiring very careful roasting.

Tellurium is often found in minerals associated with gold ores, which are usually highly auriferous or argentiferous. The presence of tellurides, when they form any appreciable proportion of the metallic minerals, usually makes an ore difficult to treat, as the action of the cyanide on the gold is slow. The cause of this slowness of action is not altogether apparent, but it is evident that there is a wide difference in the solubility of the gold in different telluride minerals and even in the same minerals from different localities. There is no action between the cyanogen radical and the tellurium, but the alkali in presence of oxygen invariably dissolves some of the metal and also acts on the sulphur when in combination, forming a solution which has a reducing action. When a telluride ore is roasted, it leaves a residue containing TeO_2, and this oxide is readily soluble in

KOH, forming a tellurite, which also acts as a reducing agent, and absorbs oxygen from the solution. The same change takes place with KCN, with the evolution of HCN. Roasted tellurides are, however, capable of being treated and the gold extracted with good results.

In cyaniding tailing from amalgamation mills, metallic mercury finds its way into the cyanide vats along with the tailing and is acted upon by the cyanide. Mercury has, however, a greater affinity for sulphur than for cyanogen, and as the solutions usually contain sulphides and sulphates, the presence of mercury is not altogether undesirable.

Zinc blende and galena have little effect on the cyanide solution.

Quantitative Analysis of the Cyanide Solution.—Place 50 grams of the ore in a wide-mouthed bottle, add 100 c.c. of KCN solution, agitate for 15 hours; filter; take 20 c.c. of solution, equivalent to 10 grams of ore, and determine its constituents. The metals in solution should be determined quantitatively by the usual methods, but for ordinary purposes a qualitative analysis is sufficient. Since alkaline suphides act so injuriously on the cyanide solution during the leaching, it is important to detect their presence. They are soluble in water. The most delicate test is by means of the nitro-prussides. They are formed by adding nitric acid to a solution of ferro or ferricyanide of potassium. Add a few drops of the nitroprusside to the cyanide solution. If any alkaline sulphide is present, even in the minutest quantity, the solution will assume a brilliant purple color.

GENERAL REMARKS

CAUSES FOR THE NON-EXTRACTION

(1). Gold alloyed or in combination.—These combinations may be broken up by a preliminary roast, or extraction effected by fine grinding, long contact with the cyanide solution, or both.

(2). Large consumption of KCN (see 'Determination of the Causes of Cyanide Consumption', page 45).

(3). Coarse gold.—This may be amalgamated or concentrated before cyaniding.

(4). Presence of soluble sulphides.—Add a soluble lead salt or an oxidizing agent.

(5). If the ore contains much kaolin or talc.—If coarse crushing is ineffectual, the separation of the sand and slime does not prove effectual, or the slime cannot be filtered or decanted, the ore is not adapted to economical cyanidation.

(6). Ores containing considerable oxidized copper are not adapted to the cyanide process, without a preliminary treatment as an acid wash. The addition of ammonium salts as NH_4Cl is usually beneficial in treating ore containing copper. The ammonia-cyanide process may be used in treating copper, nickel, and zinc ores or tailings containing values of gold and silver. Ammonia is used as a solvent for the base metals in the ore. Potassium cyanide is added to the ammonia, forming in the presence of copper, cupric cyanide ($Cu(CN)_2$) and an excess of soluble copper hydroxide ($Cu(OH)_2$), which combines the extraction of the gold and silver with the copper. The ammonia-cyanide solution is prepared by dissolving 1 to 4 pounds of KCN in a ton of solution 1% of NH_3 for each per cent of copper in the ore. With the use of open leaching tanks, a solution containing more than 1% NH_3 is not to be recommended on account of loss by volatilization, hence the excess copper should be removed by preliminary leaching with acid or water. Ammonia cupric cyanide is very stable, has a greater dissolving efficiency than ordinary cyanide, and has important oxidizing powers. This is a desirable quality in cyaniding, especially in the presence of ferrous and other reducing salts.

(7). Ores containing heavy minerals injurious to the cyanide solution.—Concentration before cyanidation may make tailing amenable to the process.

CYANIDE POISONING

For an antidote, drink ferrous sulphate followed by sodium bicarbonate. Next take a purgative. Walk about and prevent sleep. KCN in fresh cuts is liable to form painful sores.

REACTIONS IN THE CYANIDE PROCESS

IN THE PRELIMINARY TREATMENT

$$Fe_2O_3SO_3 + 6NaOH = 2Fe(OH)_3 + 3Na_2SO_4$$
$$FeSO_4 + Ca(OH)_2 = Fe(OH)_2 + CaSO_4$$
$$ZnSO_4 + 2NaOH = Zn(OH)_2 + CaSO_4$$

IN THE SOLUTION TANKS

$$4KCN + 2Au + H_2O + O = 2KAu(CN)_2 + 2KOH$$

(Ellsner's Equation)

$4KCN + 2Ag + H_2O + O = 2KCN.2AgCN + 2KOH$
(Ellsner's Equation)
$2KCN + FeSO_4 = Fe(CN)_2 + K_2SO_4$
$Fe(CN)_2 + 4KCN = K_4Fe(CN)_6$
$Fe_2(SO_4)_3 + 6KCN + 6H_2O = 2Fe(OH)_3 + 3K_2SO_4 + 6HCN$
$Al_2(SO_4)_3 + 3H_2O + 6KCN = Al_2O_3 + 3K_2SO_4 + 6HCN$

IN THE AIR

$2KCN + CO_2 + H_2O = 2HCN + K_2CO_3$
$KCN + O = KCNO$
$2KCNO + 3O = K_2CO_3 + CO_2 + 2N$
$2KCN + 2H_2O = 2HCN + 2KOH$

SUBSTANCES IN SOLUTION

$Zn(CN)_2 + 4KOH$ (excess) $= K_2O.ZnO + 2KCN + 2H_2O$
$K_2Zn(CN)_4 + Ca(OH)_2 = 2KCN + Ca(CN)_2Zn(OH)_2$

IN PRECIPITATING TANKS AND ZINC BOXES

$2KAu(CN)_2 + Zn = K_2Zn(CN)_4 + 2Au$
$2H_2O + Zn = Zn(OH)_2 + 2H$
$Zn(OH)_2 + 2KCN = Zn(CN)_2 + 2KOH$
$Zn(CN)_2 + 2KCN = K_2Zn(CN)_4$
$2KOH + CO_2 = K_2CO_3 + H_2O$
$HCN + KOH = KCN + H_2O$

Electrolysis will give from $KAu(CN)_2 = Au + K + 2CN$,

$$K + H_2O = KOH + H$$
$$2KOH + Zn = ZnK_2O_2 + 2H.$$

Report.—Make out a final report of all tests made in a neat form with all notes and observations that may be of value in the further treatment of the ore. Compare with the net returns that would result from the use of other available methods such as amalgamation, chlorination, shipping to smelter direct, concentration and shipping concentrates to the smelter.

CYANIDE TEST ON LOTS OF 30 POUNDS AND MORE OF ORE

The procedure in this case is conditioned upon the results obtained in the preliminary tests. A scheme of treatment for each ore must be devised. If the ore is to be treated by percolation the following suggestions will be of value.

Take 30 pounds of ore and crush to the required mesh, being careful to add the proper amount of alkali. Spread the ore evenly over the bottom of one of the tanks in the leaching plant, shown in Fig. 10. Ten pounds of strong solution (0.25% KCN) is run on and is immediately drawn off so that the solution just covers the ore. It is allowed to remain about twenty hours. The solution is next drained slowly off into its sump, allowing four hours. A weak solution (9 pounds of 0.05%) is next run on, a portion being drawn off immediately and run into the strong solution sump until equal in quantity to the strong solution originally applied. Contact is again allowed for twenty hours with four hours for drawing off. A water wash of 10 pounds is finally run on and in an hour or two as much is drawn off into the weak solution sump as there was weak solution added. The water wash may then be allowed

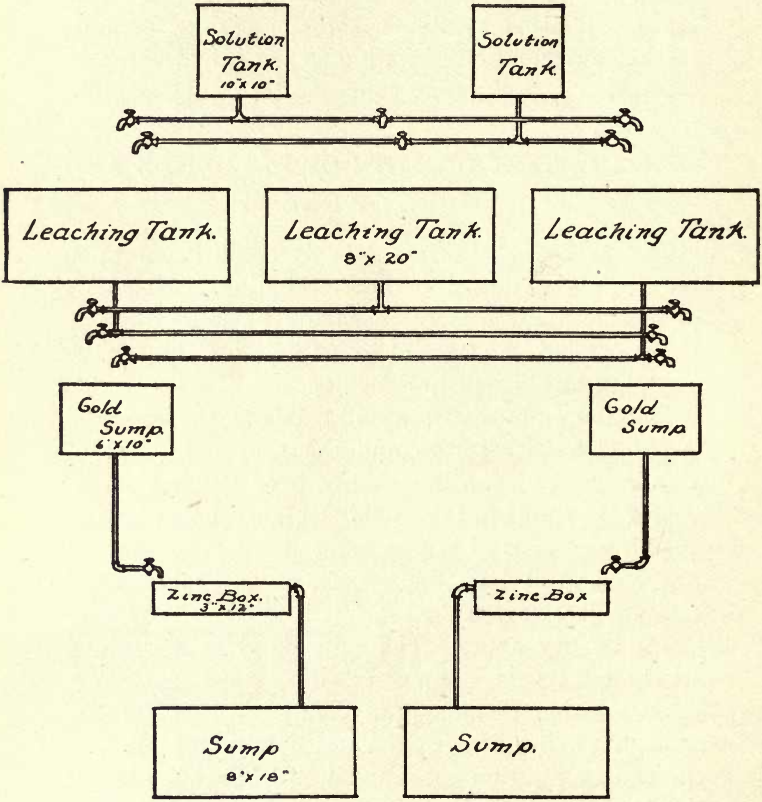

Fig. 10. LABORATORY CYANIDE PLANT

to run off immediately, but slowly, so as to take perhaps twenty-four hours from the time of applying. The ore residue is next assayed, and the sump solution passed through the zinc boxes or some other form of precipitation apparatus. If zinc shavings are used, the metal is recovered by dissolving the shavings in acid, collecting the residue on a filter, and then the whole metal value recovered as described under 'Bullion Refining.' The solutions are assayed for gold and silver, before and after passing through the zinc boxes. This will give the efficiency of the precipitation.

DOUBLE TREATMENT

A modification of cyanide process is to give part treatment in one vat, and then to transfer the material, after draining, to a second, where the leaching is completed. There are two advantages in this system of working: (1) in turning over the whole mass during transfer any closely packed unleachable portions in the first vat may be broken up and distributed among the rest of the sand; (2) the material partly treated and moistened with cyanide solution is exposed to contact with the oxygen of the air during the removal from one vat to the other. Usually in the first vat an alkaline wash, a weak cyanide solution and a medium solution or part of the strong solution, are passed through the material in order that a sensible amount of cyanide solution may be present at the time of transfer. Then, in the second vat, the remainder of the strong solution is added, followed in the usual way by the weak solution and water wash. There is no doubt whatever as to the improvement in percolating qualities caused by the transfer from one vat to the other, in the case of ordinary mixed or unclassified material; and even with the sand product of the present classification methods there is a sensible advantage in this respect, for it is found that these sands occupy a larger space, being sometimes 20% more in the second vat than in the first. As regards oxidation, there must be also some slight advantage, particularly when dealing with unclassified sand.

PRECIPITATION BY MEANS OF ZINC

When a positive metal having a difference of potential at two parts of its surface, is immersed in a salt of a less positive one, the dissolving part becomes the anode and the less soluble part the cathode; and precipitation of the less positive metal proceeds as if the current were applied from an external source. If the salt is a complex one as $KAu(CN)_2$, the action may be said to take place as follows. Let it be assumed that the precipitating metal is zinc, which contains lead as an impurity, and the solution potassium cyanide containing some $KAu(CN)_2$. Currents are generated by the dissolving action of the zinc and flow through the solution from the zinc to the lead. In the paths of these currents, molecules of $KAu(CN)_2$ are carried by diffusion and their ions are directed, the $Au(CN)_2$ going to the zinc and the K to the lead, where they simultaneously give up their charges and become atoms or molecules. The gold deposits on the part of the zinc where the current enters and the $(CN)_2$ is left free to combine with KOH or water. The K ions give up their charge in the same way as the lead and becomes an atom forming for the moment an anode, while the lead is the cathode. Gold is then deposited on the lead as the solution diffuses to that electrode. Thus, when a zinc-lead couple is employed, gold deposits both at the zinc and the lead pole, whereas when we apply a current from an external source, gold is deposited permanently only at the cathode. At the moment the atom of gold is deposited on the zinc its tendency is to redissolve, but a new positive or anode part of the zinc takes this atom as a negative electrode or cathode on which to deposit potassium atoms. The potassium, as it separates, keeps up local currents that precipitate further quantities of gold as the solution diffuses, and thus the deposit thickens. The proportion of gold actually precipitated by the primary action of the zinc must necessarily be small, as the surface soon becomes covered with gold, and the chief amount must be obtained by the secondary action of the potassium. In the case of precipitation from very weak solutions as those used in the treatment of slime, the use of the zinc-lead couple obtained by dipping the zinc shavings in lead acetate will give very satisfactory results. With strong solutions the precipitation by zinc alone is efficient, and the use of the couple is not necessary. With zinc shavings alone, all authorities are of the opinion that strong solutions give uniformly good results, while weak solutions give results that are more or less erratic.

Consumption of Zinc.—The actual amount of zinc dissolved during the precipitation is a matter that is not easy to determine, but it is certainly many times in excess of the theoretical quantity required. If the whole of the zinc dissolved were used to precipitate the gold or silver in the solution, 1 ounce should deposit 6 of gold or 3 of silver, and this is about what would be obtained in a concentrated solution of $KAu(CN)_2$ or $KAg(CN)_2$. In actual practice it requires from seven to ten times the theoretical amount of zinc, which for the most part is wasted in decomposing water and liberating hydrogen.

Zinc Box Operations.—The gold-bearing solution is allowed to flow through the box at as uniform a rate as practicable. The most suitable rate of flow, which varies with the sectional area of the column of zinc, with the area of the zinc surface, and with other factors, is found by trial. With a little practical experience the operator gets a rough general idea as to whether precipitation is effective or not. If the precipitate is loose, of a black or dark brownish color, it may be taken for granted that the gold is precipitating satisfactorily. If, on the other hand, the deposit is firm and light colored, it is probable that the sump solutions will assay high. It is usual then to take the precaution of adding a few pounds of cyanide in lumps or of caustic alkali to the head of the zinc box and thus temporarily increase the action of the zinc. The light color then disappears and the deposit should become loose. The light-colored precipitate is not always a sign of poor precipitation, as it may be caused by aluminous or other matter precipitated from the solution, but it is generally advisable to keep any such precipitates at a minimum. If this cannot be done by temporarily increasing the strength of the solution, it is advisable to clean up often.

Copper in Solution.—Much difficulty has often been experienced by the presence of copper in solution. Zinc in a cyanide solution precipitates copper, but the potential difference between the two metals is small compared with that between zinc and gold, or zinc and silver. The effect of this is that a solution which would just precipitate a loose black deposit of gold would not be at all suitable for throwing down a similar mass of copper. If copper were present in such a solution it would be precipitated as a firm metallic coating, such as would raise the decomposition point of the solution above that which the potential of the zinc was capable of effecting, and then the action would fall off. Should the solution be sufficiently strong in salts to deposit the copper in a loose form, both gold and copper precipitate together; but as the solution pressure of the copper is higher than that of gold, a larger porportion of the former is re-dissolved and the net result is a poorer precipitation. The strength of solution required to produce this loose deposit of copper is not often met in present treatment of gold ores, and it is therefore advisable, from an economic point of view, when copper is troublesome, to resort to other methods of precipitation, such as the electrical process. When copper is known to be present in the ore, care should be taken to make exhaustive quantitative tests before deciding to employ zinc as a precipitant.

Electrical Precipitation.—When an electric current is applied to a cyanide solution containing gold or silver, the K ions travel toward the cathode and the $Au(CN)_2$ or $Ag(CN)_2$ toward the anode. Hence it is important to overbalance this influence of the current by giving motion to the solution so as to bring the gold and silver-bearing ions in contact with the cathode. Some of the solution, however, must get to the anode and is disassociated as atoms, and this accounts for the gold found with the ferric oxide attached to the iron plates when they are used as anodes.

It has been found that to obtain a good adherent deposit in electroplating, the current must be roughly proportional to the quantity of gold per unit of solution. For a current of 10 amperes per square foot, the poorest solution used contains 50 ounces of gold per ton, and this is equivalent to 0.01 ampere per square foot for each 0.05 ounces of gold in solution.

Siemens & Halske Process.—In this process the anodes are of sheet-iron from ⅛ to ¼ inch in thickness, sewed up in covers of hessian or other textile material to prevent short circuiting. The plates are connected to the positive pole of the dynamo. The cathodes consist of lead foil 0.0032 inch thick, weighing about 0.19 pound per square foot. These are connected to the negative pole of the dynamo. The average current used is 0.04 amperes per square foot of anode with a cathode surface of 50 to 150 square feet per ton of solution per 24 hours, according to the conditions.

Butters Process.—This differs from the Siemens & Halske process in that insoluble anodes of peroxidized lead are used. They do away entirely with the sludge due to the dissolution of iron anodes. They also make it possible, by the use of a high current density, to precipitate the bullion as a pulverulent deposit which can be recovered by wiping the cathode, thus obviating the necessity of destroying the cathode to separate the metallic deposit, as is necessary in the Siemens & Halske. The ordinary tin plate of commerce is used as a cathode, and the current density of 0.28 to 0.56 amperes per square foot of anode is used.

TREATMENT OF CONCENTRATE

The cyanide process is now applied in many cases to materials which, a few years ago, would without question have been treated by roasting and subsequent chlorination. This is true especially of products of close concentration, which are often cyanided raw, but, if rich enough, are preferably roasted. Cyanidation was at first applied to these refractory materials only, which were then treated by agitation, alternating with percolation. From a purely technical point of view, this is undoubtedly the quickest and most efficient method of extracting the gold from such material, whether raw or roasted. But the expense of continuous or repeated agitation of coarse particles of high specific gravity limits the application of this method to rich material. The general practice is to crush the whole of the material very fine and to treat it as slime by agitation and filter pressing.

In treating slime by cyanidation there are two main difficulties: (1) that due to the viscosity of the pulp which diminishes the dissolving power of the solvent for gold and silver; (2) due to the presence of substances capable of combining with absorbed oxygen. To overcome these difficulties the viscosity of the pulp is diminished, air is supplied from an external source, and the pulp kept agitated while in the vats.

SLIMING TESTS FOR 5 TO 20 POUNDS OF ORE

Preparation.—The sample of the ore is ground in the coffee mill to 30 or 40 mesh and then introduced into the pan grinders used for silver amalgamation where it is made into a thick pulp with water and reduced to the required fineness. Lime is added as previously determined. A dilute cyanide solution of 0.005 to 0.10% strength is generally used instead of water. Use approximately one pound of solution per pound of ore. The slime is agitated in an apparatus shown in Fig. 11. When agitating slime alone use six parts of solution to one part of slime. When both sand and slime are present, a thicker pulp must be used according to percentage of slime present.

Use of Vacuum Leaf Slime Filter.—Run the slime into the filter-tank when the agitation is finished and start the air pump connected with the filter-frame. Note the vacuum and the time required to build up a cake of slime on the filter, one inch thick. Always keep the filter-frame covered with solution. When the required slime cake has been built up,

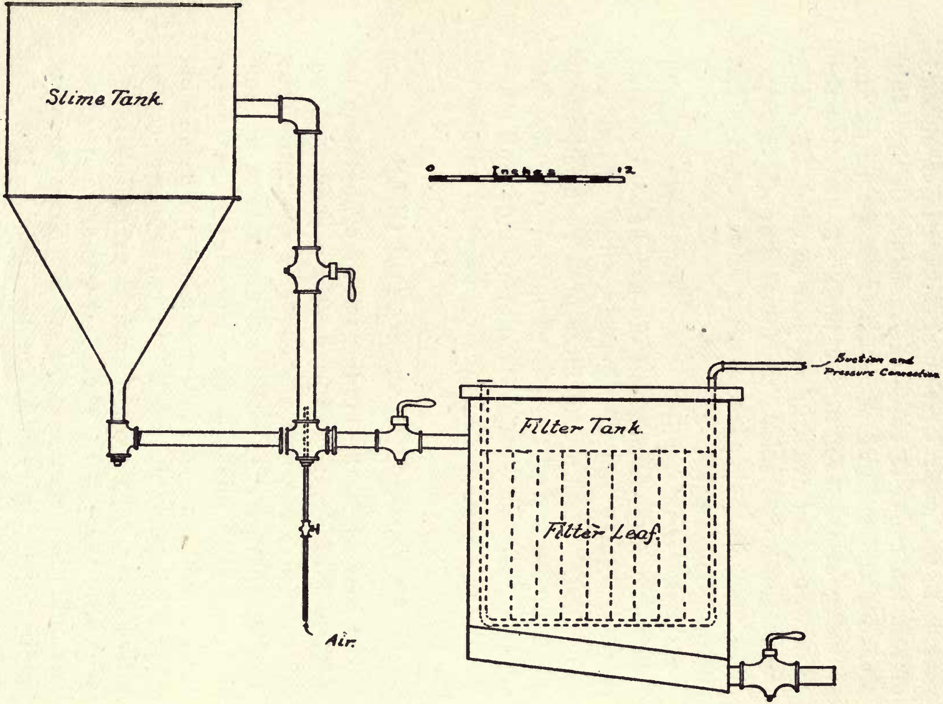

Fig. 11. SLIME AGITATOR AND LEAF FILTER

remove to the weak solution and draw in an amount of solution equal to twice the moisture in the cake. Measure the amount of solution drawn in by the lowering of its level in the tank. Remove to the wash water and draw in one-half as much as of the weak solution. Remove the filter, shut off the vacuum, turn on the water pressure and discharge the adhering slime cake.

Note the time required for these operations, the area of the filter-frame, and the weight of dry slime treated. Assay the discharged tailing. From the above data compute the filter area required for a given tonnage (one ton per 24 hours), the percentage of extraction, consumption of cyanide, and other calculations pertaining to the commercial success of the test.

CYANIDATION OF SILVER ORES

The success of the application of cyanidation to treatment of silver ores is greatly dependent upon the condition in which the silver exists in the ore; that is, whether it is soluble or insoluble or in combinations with minerals that have injurious effects upon cyaniding. The forms of silver practically insoluble or sparingly soluble in cyanide are, coarse native silver, ruby silver (Ag_3AsS_3 or $AgSbS_3$), and stephanite ($AgSbS_4$). These last two are more readily soluble in mercuric cyanide. The silver minerals readily soluble in cyanide are, the chloride, bromide, and chloro-bromide of silver. The tests for cyanidation of silver ore are practically the same as with a gold ore with the following exceptions; the time of treatment is usually greatly lengthened, stronger solutions are required (0.25 to 0.75% KCN), sliming is generally necessary with a thorough and prolonged agitation and oxygenation of the pulp. A greater consumption of cyanide must be expected on account of the larger amount of metal extracted as compared with gold. Concentration may often precede the cyanide treatment to advantage. Rebellious ores might be given a chloridizing roast, as a plain roast invariably interferes very seriously with silver extraction. The increased extraction due to the roast must warrant its high cost and loss of silver.

CHLORIDIZING ROAST OF SILVER ORES

Principles and Reactions.—Chloridizing roasting is, generally speaking, a necessary preliminary to the treatment by wet processes (amalgamation or lixiviation) of all the silver ores in which that metal occurs as a sulphide, sulphantimonide, or a sulpharsenide, as well as those in which silver sulphide occurs not merely mingled but isomorphously associated with galena, blende, or other sulphides. Ores for chloridizing require to be crushed from 16 to 40 mesh preparatory to roasting. When the silver-bearing mineral is brittle and decrepitates easily with heat, the ore only requires crushing to 16 mesh, but dense ores which do not have this property, especially those containing much blende or galena, require to be crushed to 40 mesh to secure good results. The chloridizing roast aims to convert as much as possible of the silver present into chloride, and of the other metals into their oxides; the agency employed being common salt.

The ores are usually divided into three groups: (1) those containing more than 8% sulphur, which usually require a preliminary roast before adding the salt; (2) ores containing 3 to 8% sulphur; (3) ores containing less than 3% sulphur. These usually require the addition of sulphur, as iron pyrite, in order to decompose the salt and liberate the chlorine.

The percentage of salt used varies from 10 to 15%

and can be determined best by trial. With some ores it is found better to add salt at the beginning; with others near the end of the roast. Ores containing no As, Sb, Pb, or Ca are generally best treated by adding salt at the beginning. If arsenic volatilizes as a sulphide the loss of silver seems to be less than when it volatilizes as a chloride. Lead and lime should, if possible, be kept as sulphates, for otherwise they are great consumers of chlorine. If an ore carries sulphides of lead, zinc, copper, and iron, it is roasted at a just visible red, in order to form sulphates. Sulphates of iron and copper will decompose salt with the liberation of chlorine and the other two will not.

$$2NaCl + 2FeSO_4 + 3O = 4Cl + 2Na_2SO_4 + Fe_2O_3.$$

It is therefore advisable to form $FeSO_4$, $CuSO_4$, and as much sulphate and oxide of lead and zinc as possible, and yet not decompose the $FeSO_4$ and the $CuSO_4$. For this reason the roast is carried on slowly and at a low temperature until the required stage is reached. The salt may then be added. The roast is ordinarily finished at a cherry-red heat. The chlorine liberated together with HCl formed will chloridize the ore, as below:

$$\text{Steam } (H_2O) + CuCl_2 = CuO + 2HCl$$
$$2HCl + Ag_2S = 2AgCl + H_2S$$
$$8Cl + Ag_2S + 4H_2O = 2AgCl + H_2SO_4 + 6HCl$$

Careful note should be taken of the duration of the roast, heat used, and at what time the salt was added.

Testing an Ore.—Crush the ore so as to pass through a 30 or 40-mesh screen. Sample carefully and determine the total gold and silver content. Weigh out from 300 to 500 grams of the ore and roast in an iron dish in the gas oven furnace. The time of adding the salt and the amount used is determined by the sulphur content of the ore as previously given. Roast slowly and carefully (from 30 minutes to two hours), with a frequent stirring to prevent caking, and finish at a cherry-red heat. In practice the chlorination is often increased by withdrawing from the furnace and allowing the ore to stand on a cooling-floor from 24 to 60 hours and then moistening the cooling heaps with water, by which means an additional chlorination of 3 to 6% is often reached.

After roasting weigh carefully to determine loss and then pass through a 30-mesh screen. Sample carefully, take enough ore from the following tests, and grind it in an agate or porcelain mortar so as to pass through a 100-mesh screen. If iron receptacles were used for grinding the following reaction might occur: $2AgCl + Fe = FeCl_2 + 2Ag$. The result would be material not readily soluble in hyposulphite solution.

Take 5 grams and assay. Calculate the total weight of gold and silver in the roasted ore, then the difference between this and the total amount in the raw ore equals the gold and silver loss during the roast.*

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*See 'Reducing Factors' for reason for calculating losses on total weights of silver and gold instead of difference of assay (p. 88).

Soluble Salts.—(Excess of NaCl used and all chlorides and sulphates soluble in water, including AgCl soluble in the NaCl solution, if an excess has been used in the roast). Weigh out 5 grams of the roasted ore and leach with hot water by decanting through a filter paper. Wash the ore into the filter with hot water. Dry, burn off the filter paper and weigh. Loss in weight equals soluble salts. As the percentage of NaCl used in the roast increases, the soluble salts generally become more abundant.

Silver as Sulphate and Silver Salts Soluble in Water Containing Salt (NaCl).—Assay the whole residue after leaching with water. The difference between this weight and the weight of the button from the 5 grams of roasted ore equals the silver salts soluble in water or brine solution. Any undecomposed salt left in the ore will precipitate the $AgSO_4$ as chloride.

Silver Salts Soluble in Hyposulphite.—Place 15 grams of the roasted ore in a beaker, add 300 c.c. of hot water, decant the water through a filter; add 250 c.c. of 5% hypo solution ($Na_2S_2O_3 + 5H_2O$), allow to stand on ore for one-half hour at 125°F., stirring frequently.

$$2AgCl + 2Na_2S_2O_3 = 2NaCl + 2NaAgS_2O_3$$

Decant several times with hot water through the same filter as first used and finally wash with a little fresh hypo. Dry the residue, weigh and assay the whole of it. The difference between one-third the weight of the button obtained and the assay of the roasted ore equals the silver salts soluble in water and hypo. Then subtract the silver salts soluble in water and get silver salts soluble in hypo alone. If the tailing contains a quantity of gold that will pay to extract, a weak solution of cyanide may be used effectively in the same apparatus by giving the ore a thorough water wash following the hypo lixiviation and preceding an application of cyanide solution. The gold is recovered from the cyanide solution in the usual way.

Report.—Make out a tabulated report of results obtained together with cost data. Compare with cost and net returns by shipping direct to smelter.

PAN AMALGAMATION FOR SILVER ORES

Principles.—Amalgamation of ores in pans is used for the recovery of silver from free milling ores, those in which the silver is in the native state or in the form of a chloride or bromide; a small percentage of the sulphide (argentite) is allowable. If the ore is not free milling, it must first be given a chloridizing roast, as already described, to change the silver-bearing sulphides, arsenides and antimonites to chlorides. The chief object in pan amalgamation is to keep the particles of ore for a sufficient length of time, in contact with the finely divided mercury globules with or without the presence of certain reagents to aid the amalgamation. The pan is used instead of plates as the ore can be kept in contact with the mercury for a much longer time and facilitates the use of heat and chemicals.

Interference of Various Substances.—Any substance which tends to make the mercury dirty or coat the fine globules will both interfere with the amalgamation and increase the loss of mercury. Grease in any form, as in exhaust steam, is fatal to good results, while talc, kaolin, and other hydrated silicates of magnesia and alumina act similarly by coating the globules of mercury and preventing contact between them. Lead ores interfere by 'sickening' the mercury. Under favorable circumstances, the lead ores, when in the chloride form, are rapidly amalgamated, forming a pasty, rapidly tarnished amalgam, which is very easily floured and thus causes

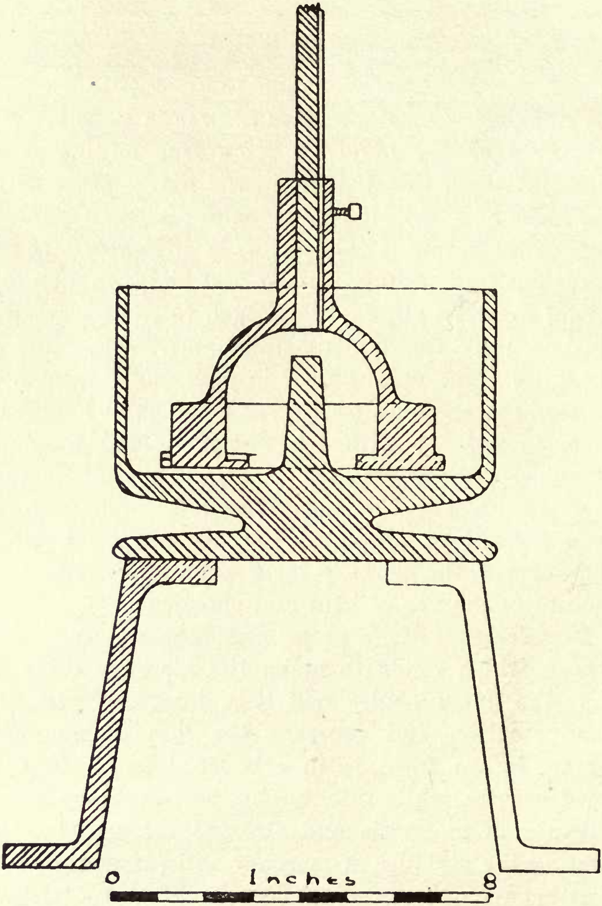

Fig. 12. PAN AMALGAMATOR AND GRINDER

a large loss of mercury. The lead chloride may be changed to the sulphate, which will not readily amalgamate, by the addition of copper sulphate to the charge, as below:

$$PbCl_2 + CuSO_4 = PbSO_4 + CuCl_2$$

Arsenical and antimonial ores are always difficult to work as the mercury not only becomes 'floured' but also 'sick' and 'dirty,' so that it will not readily amalgamate. In this case the whole stock of mercury should be frequently cleaned with nitric acid and retorted. Manganese oxides froth in the pan and give rise to a high loss of mercury and a low percentage of extraction.

Addition of Chemicals.—In the case of some free-milling chloride ores, no additions of chemicals need be made, but the use of salt has the effect of shortening the process. In the case of all refractory ores, the yield is increased by the addition of both salt and copper sulphate. When the zinc blende and pyrite are present and the gangue is chalcareous, some authorities claim that salt and bluestone should not be added. Lime consumes bluestone, forming calcium sulphate, and the sulphides seem to flour the mercury in the presence of salt and perhaps form chlorides. With special ores, other additions of chemicals are often made. Thus with ores containing oxide or carbonate of copper, sulphuric acid in the proportion of one to two pounds per ton may advantageously replace all or part of the copper sulphate. Lime in the proportion of one to one-and-one-half pounds per ton is frequently employed with ores containing partly oxidized pyrite. Iron borings in quantities up to 20 lb. per ton are frequent additions to such ores as corrode the pan rapidly.

Chemical Reactions.—Many reactions take place that may be due to the constituents of the ore or may arise from chemicals. The following are some of the probable reactions:

$$CuSO_4 + 2NaCl = CuCl_2 + Na_2SO_4$$
$$CuCl_2 + Ag_2S = 2AgCl + CuS$$
$$CuO + H_2SO_4 = CuSO_4 + H_2O$$
$$Hg_2Cl_2 + Fe = FeCl_2 + 2Hg$$
$$2AgCl + Fe = FeCl_2 + Ag$$
$$Ag_2S + 2Hg = Ag_2Hg + HgS$$

Procedure for Experiment.—Clean all rust and dirt from the pan and mullers as it is essential to good results that the pan bottoms, mullers, and dies should be free from graphite or iron rust. Pour water into the pan to the depth of one inch and set the mullers in motion. Now slowly add three or four pounds of the ore which has been previously sampled, assayed and weighed. Add a little water from time to time to keep the mass in a pasty condition. The charge should finally be about the consistence of a thick paint. Care should be taken not to have the pulp too thin or the mercury, when added, will sink to the bottom and not come thoroughly in contact with all the ore particles. Add the chemicals suitable for the ore selected. The following may be taken as an example: salt 3½%; copper sulphate 0.1 to 0.15 per cent.

Allow the mullers to rest on the dies and grind for one-half hour or until all the lumps have been ground and the ore is finely divided. Now raise the mullers one-half inch from the dies and add 5 to 10% mercury (weighed), spraying it through a chamois skin on top of the pulp. If the pulp is of the right consistence, the mercury will be found disseminated in fine globules throughout the mass. With the aid of a Bunsen burner, heat the pulp to a temperature of 175 to 185° F. and maintain this temperature until the end of the run.

At the end of two hours, thin out the pulp with water and continue the agitation for 15 minutes to settle out the mercury. Pour off all the overlying pulp possible and separate the mercury and amalgam from the remainder of the tailing by panning or by using a hydraulic classifier. The tailing should be sampled and assayed for gold and silver. The silver is recovered from the amalgam as described under 'Retorting.' Weigh the silver and mercury recovered.

Report.—(1) Extraction based on assay of heads and tailing; (2) extraction based on actual recovery; (3) loss of mercury in pounds, per ton of ore; (4) itemized cost of the operation.

POT ROASTING

General Principles.—'Lime roasting' is a term proposed by W. R. Ingalls for the operation of forcing air under pressure through a mixture of galena and lime at the kindling temperature, with the object of oxidizing the lead and the sulphur and of fritting or fusing the charge. It may be considered as a limitation of the broader term of pot roasting which does not specify the use of lime or restrict the process to galena alone.

At the present time there are two views as to what reactions take place during the process, but the evidence and available data are too incomplete to enable a definite conclusion to be reached. Some facts, may, however, be stated. There is clearly a reaction to a certain extent between lead sulphide and lead sulphate as in the reverberatory furnace, because prills of metallic lead are to be observed in the lime-roasted charge. There is a formation of sulphuric acid in the lime roasting, upon the oxidizing effect of which Salvelsberg lays considerable stress, since its action is to be observed on the iron work in which it condenses. Calcium sulphate, which is present in all the process, being specifically added in the Carmichael-Bradford, evidently plays an important chemical part, because not only is the sulphur trioxide expelled from the artificial gypsum, but it is also to a certain extent expelled from the natural gypsum in the Carmichael-Bradford process. Further evidence that lime does indeed play a chemical part in the reaction is presented by the phenomena of lime roasting in clay dishes in the assay muffle, wherein the air is not blown through the charge, which is simply exposed to superficial oxidation in ordinary roasting. The desulphurized charge dropped from the pot is much below the temperature of fusion, while even in the interior pyrite and zinc blende are completely oxidized. This at least, indicates atmospheric action.

Experimental Work.—Take 20 lb. of galena concentrate and crush through a 10-mesh screen. Sample carefully and assay for FeO, CaO, MgO, SiO_2 and S. Make up a charge of ore 20 lb., with limestone or quartz as required to make a singulo-silicate slag, and enough water to make the whole charge of the consistence of brasque. The roasting pot is now warmed by burning a layer of charcoal on the grate, with a gentle air-blast. A layer of coarse limestone is placed over the grating as a safeguard, then followed by a thin layer of ignited charcoal and the blast turned on to about two inches water pressure. As soon as the charcoal is burning freely it is covered with a second layer of limestone, pea size. The charge, which should be thoroughly mixed, is now fed gradually. A thin layer is spread over the limestone and as soon as it becomes ignited and glowing spots begin to appear on the surface, a second layer is added and so on until the pot is filled. The hot junction of a thermo-electric pyrometer is buried half way in the charge and readings are taken every minute. The blast-pressure is now raised to about six inches of water and held until no more fumes pass off from the surface. Any new holes that form should be immediately filled by poking down

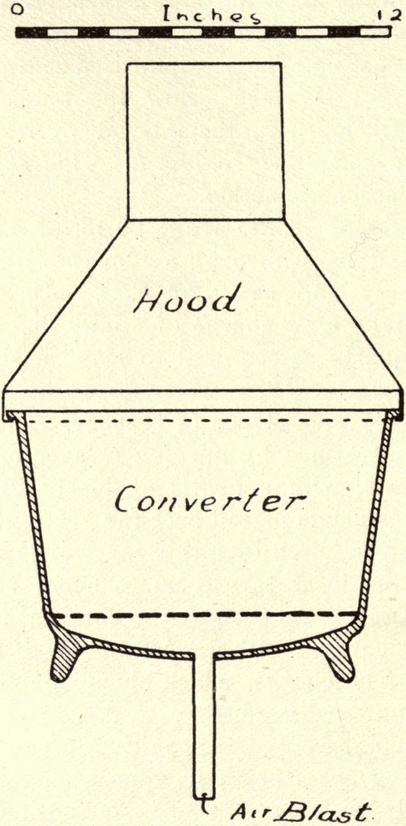

Fig. 13. POT ROASTER

the charge into the cavities. When the charge is cooled down somewhat, it is dumped out. The coarse limestone is picked off the bottom, the slagged and unslagged portions are separated and weighed; the slagged portion is crushed and added to the pulverulent part, the whole sampled down and assayed for gold, silver, lead and sulphur.

The conditions of the test may be varied to show the effect of adding more or less limestone or other flux, high or low blast, on the loss of lead, silver and the elimination of sulphur. This roast may also be used for sulphide ores other than galena, such as pyrite and chalcopyrite.

Report the losses of lead, gold, silver, and sulphur in the roast. Loss in weight of the charge. Percentage of slagged and unslagged material. Time of roast, temperatures and blast pressures. Cost data.

PARKES PROCESS

General Principles.—The Parkes process is based on two facts (1) the greater affinity of silver for zinc than for lead; and (2) the insolubility of zinc-silver alloys in lead which is already saturated with zinc. The process of desilverization consists, in brief, of stirring 1 to 2% of zinc into a bath of molten work-lead or base-bullion heated to above the melting point of zinc (415°C.), and allowing it to cool, when a crust or scum rises to the surface, containing nearly all the silver. A repetition of the process with a smaller amount of zinc leaves a lead which is practically free from silver, or not containing more than 0.2 to 0.3 oz. per ton.

Zinc has an even greater affinity for gold and copper, and copper alloys so readily with zinc, it is important to purify the bullion as much as possible by softening and drossing before the desilverization is commenced; otherwise a poor copper crust is formed before silver can be extracted, thus wasting zinc as well as time. It is noticeable that a gold-copper crust containing all but the minutest trace of gold present, is formed long before the lead begins to take up the zinc to any considerable extent, the gold-zinc and copper-zinc alloys being apparently almost insoluble in zinc-free lead, whereas it is impossible to produce a silver crust without first saturating the lead at the given temperatures. Bismuth does not interfere with desilverization,* but antimony in the proportion of 0.1% and arsenic in even smaller proportions, not only retard the rising of the crusts but even prevent a clean separation from the underlying lead. On a commercial scale the antimony is removed by drossing at a bright red heat and the arsenic is oxidized by softening 8 or 9 hr. at a yellow heat. Nickel and cobalt, rarely present, enter the crust like copper but both are removed in the dross.

All crusts are separated as far as possible from adhering lead by liquation before being further treated. This may be done in furnaces or in pots. The liquation pot, of whatever form, must be heated very gradually, otherwise some of the zinc crust will be dissolved in the liquated lead. When no more lead exudes, the crust is withdrawn and broken up for retorting off the zinc. No matter how carefully this be done the liquated lead will contain 30 or 40% more silver than the desilverized lead. In practice it is put back into the next charge entering the zinking kettles.

Procedure.—Take about 20 lb. of lead, sample and assay, melt down in a crucible very slowly. When the lead is finally melted, skim off the dross, being careful to keep the lead just above the melting point. This dross or skimming will contain nearly all the copper, if the melt has been slow enough. Raise the

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*Bismuth is an impurity detrimental to 'corroding' market lead. This impurity is not eliminated in the Parkes process, but it may be removed by either the Pattison or the Betts electrolytic process.

temperature and keep the material at a good red heat until a dross forms. The lead is to be tested from time to time until the antimony is removed as shown by the appearance of a sample. Before the antimony has been removed, a sample of the bullion taken in a ladle will work; that is, small particles of black melted skimmings will float on the surface of the lead, with a rotary motion which resembles that of particles of grease on hot water. As the softening approaches the finishing point the globules become less in number and size, a coating of yellow litharge forms more readily on the molten red hot lead, and finally no more globules are seen and litharge forms quickly. When the antimony has been removed, skim off the dross formed during the operation.

Raise to a yellow heat and cook for some time with a free access of air to oxidize the arsenic. Pour a sample of lead into a mold and allow to cool slowly, skimming with a flat wooden stick. If there is no more arsenic and antimony, the surface of the lead, upon solidifying, will assume a rich indigo-blue color, can be easily scratched with the fingernail, and will show a brilliant lustre on a fresh incision. When arsenic has been eliminated, cool down to a red heat and again dross. Weigh each dross as removed.

Cool down to a little above the melting point of zinc (415°C.) so the stick of wood will just be ignited by the lead. Take sample No. 1 for assaying by dipping out with a scorifying dish held in a pair of tongs. It is very important, when adding zinc, to have the temperature of the lead *only above the melting point of* *zinc*, otherwise the zinc will oxidize and prevent the operation being successful.*

For the **first zinking** add 6 oz. of zinc and stir in well with a stick of wood, cool down until the lead is just molten and thoroughly dross off the zinc crust that rises to the surface. This dross will contain most of the silver and all of the gold. Heat up again to a little above 415°C. and take sample No. 2. In the **second zinking** add 4 oz. of zinc and proceed as in the first zinking. Reheat, stir, and take sample No. 3. For the **third zinking** add 2 oz. and proceed as before. Take sample No. 4. After the last zinking, the lead is again raised to a good yellow heat until any remaining zinc is driven off. After this is accomplished, skim and pour into a heated mold, allow to cool and weigh.

In treating the zinc crusts take all of them, place in a crucible (See Fig. 14), heat very slowly to the melting point of lead, and allow the excess lead to liquate off slowly. Gradually increase the temperature so that all the free-lead will drain off (in the meanwhile agitating the crusts with a rod so as to break up large pieces and help work off the lead), *but do not heat so high that the zinc will distill or oxidize.* Weigh and assay the liquated lead. The dry zinc crusts are placed in a retort and heated in a gas forge and the zinc fumes caught in a condenser. When all the zinc is distilled, the remaining 'rich-lead' is poured into a bullion mold, and the silver and gold

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*Before zinking it is well to add a small strip of aluminum and stir into the lead. This will prevent oxidation during the operation.

recovered by cupelling off the lead in a large bone ash cupel. If the lead contains too much zinc it will not readily cupel and the samples should first be scorifed.

Report.—Weigh up the silver and gold recovered, assay samples and note the amount of silver and gold left after each drossing. Account for all the gold and silver which in 20 lb. of base bullion should be equal to: (1) amount of Au + Ag recovered; plus (2) amount left in the lead; plus (3) amount of losses due to cupellation, etc. The latter may be determined by difference between Au + Ag button actu-

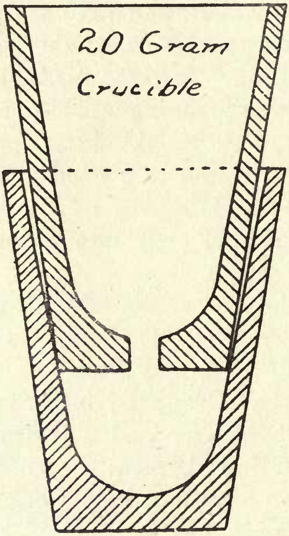

Fig. 14. LIQUATOR FOR ZINC CRUSTS

ally recovered and the calculated recovery. Calculate the percentage of recovery, cost and the profit or loss based on the recovery. Give weight of original lead ingot, weight of the liquated lead, amount of zinc added and weight of dross taken from the lead, with per cent of dross as compared with the weight of the base bullion taken.

ZIERVOGEL PROCESS OR THE SULPHATIZING ROASTING OF COPPER MATTE

THE GENERAL PROCESS

Principles.—This very simple, though delicate, process depends upon the fact that the silver in copper mattes is converted into silver sulphate by means of the sulphuric anhydride evolved from the decomposition of iron sulphate and copper sulphate, which are formed during the roasting from the iron and copper sulphides present in the mattes. The process consists of roasting the sulphides with an excess of air, so that they are oxidized partly to sulphates, which are then heated sufficiently to decompose the sulphates of iron and most of the sulphates of copper, while the silver remains sulphatized as $AgSO_4$ and may be dissolved out with boiling water to be subsequently precipitated on metallic copper. Simple as it appears this process is extremely difficult to execute, for it requires a high degree of skill to seize the exact period when the iron and copper sulphates are converted into their insoluble higher oxides and when none of the silver sulphate is decomposed.

Procedure.—Grind about 1½ lb. of matte so as to pass through 60 mesh. Take sample No. 1 and assay for gold, silver, iron, copper and sulphur. Carefully note the amount taken for the roast. Put the matte in a cast-iron roasting dish and place on the hearth of the gas-oven furnace which has previously been lighted. Heat up to 325°C., as indicated by a thermo-electric pyrometer. The hot junction should be placed at the surface of the ore. At this stage the sulphur of the matte begins to burn and the temperature will then rise rapidly owing to the heat of combustion derived from the burning of the sulphur in the matte. ($S + 2O = SO_2 + 71000$ calories.) Stir continuously to prevent caking. Roast for about three-quarters of an hour between 450 and 550°C., reaching the latter temperature at the end of the period. By this time the major portion of the sulphur will have been oxidized. Take sample No. 2 with a sampling spoon.

Now the heat derived from the combustion of the sulphur will decline and the gas and blast must be increased. Roast for 1½ hr., gradually increasing the temperature to 690°C. at the end of this time. During this period the sulphates will be formed. Take sample No. 3.

The composition of matte is Cu_2S, FeS, Ag_2S.

At 570° C.;

$$FeS + 3O = FeO + SO_2.$$
$$SO_2 + O = SO_3.$$
$$FeO + SO_3 = FeSO_4.$$
$$Cu_2S + 4O = 2CuO + SO.$$
$$CuO + SO_3 = CuSO_4.$$

The formation of the copper sulphate is a gradual one and is not dependent upon the action of the gaseous products of the dissociation of the sulphate of iron.

$$FeSO_4 + \text{heat} = FeO + SO_3.$$
$$CuO + SO_3 = CuSO_4.$$

This is shown by the roasting of an iron-free copper matte in which the rapid accumulation of the copper sulphate at temperatures corresponding to those at which this compound appears in ordinary matte, proves that iron sulphate was not essential to the sulphatizing of the copper compounds.

Now raise the temperature gradually till at the end of 35 to 40 minutes the temperature reaches not higher than 845° C., at which temperature the maximum silver sulphate will be formed. Above 850° C. it will be rapidly broken up. Take sample No. 4.

At the end of this roast the sample should show little copper sulphate and no cuprous oxide because the latter will precipitate the silver from the silver sulphate in the form of bright metallic scales. This lower oxide, if present, must be re-oxidized by continuing the roasting with an excess of air before the product is ready for leaching. Test by throwing a sample into a beaker of hot water and when no bright spangles of silver appear the roast is complete.

Now raise the temperature to 900° C. in the next 15 minutes. Take sample No. 5. This sample will show the loss of silver sulphate due to overheating.

Assay the samples No. 2 to 5 for copper, iron, and sulphates, also total silver content.

Reducing Factors.—The above results are not in a condition for direct comparison. During the roasting operation much oxygen from the air was taken up in the formation of the sulphates without a corresponding loss of sulphur. The weight of the charge increases as the sulphates continue to rise in amount, and as these compounds are decomposed. The volume also increases and diminishes, as may be readily seen in the roasting dish in the furnace. In analyzing the samples taken at successive stages of the process, the same weight of each sample was taken. Because of the rapidly changing weight of the roasting charge, the amount of copper, iron or silver sulphates in these equal weights does not represent the relative amounts of these compounds in the charge in these stages of the process. To permit comparisons to be made it is necessary to ascertain quantitatively the weight of the charge.

The total amount of the copper will not change during the operation. The amount determined in equal weights of the respective samples will vary through considerable limits. From the varying percentages of copper present in the different samples the change in weight of the charge is ascertained. For a decrease in percentage of copper an increase in weight of the charge must have taken place, and for an increase in the copper content of a sample, a diminution in the weight of the charge. In other words, the relative weights of the charge at different stages are in inverse ratio to the copper content of equal weights of sample taken at these stages. Putting the weight of the original matte, before roasting commenced, as unity, the weight at any step in the operation may be obtained as follows:

I : X :: % Cu in sample : % Cu in original matte.

Here X = the ratio of the weight of the charge at the time the sample was taken to the weight before the roasting commenced. By multiplying the percentages of the different sulphates in each sample, as obtained by analysis, by the reducing factor for that sample the percentages which represent relative amounts of these sulphates in the entire charge at the time the sample was taken may be obtained. A table of corrected results may be thus constructed.

METHOD OF ANALYSIS

Total Copper.—Take a one-gram sample and treat with 7 c.c. nitric acid and 5 c.c. of sulphuric acid in a flat-bottomed flask, until white fumes come off. Cool, dilute with 50 c.c. of distilled water, transfer to a graduated flask, add 15 c.c. of ammonia, dilute to a known volume, by preference 100 c.c., allow the precipitate of hydroxide of iron to settle. By means of a pipette take off an aliquot portion of the supernatent blue solution, free from iron precipitate, and titrate for copper with a standard solution of potassium cyanide.

Soluble Sulphates of Iron and Copper.—Weigh out a 2-gram sample, place in a No. 3 beaker with 150 c.c. of distilled water. Heat on a hot plate and boil for a few minutes. Filter into a No. 4 beaker, wash the residue by decanting several times with hot water through the filter. Add a strip of aluminum two inches square and one-sixteenth inch thick, and a few drops of sulphuric acid. Boil on a hot plate till all the copper is precipitated and the iron is reduced, using a drop of K_4FeCy on a spot plate as an indicator. Decant and wash through a filter. Titrate the filtrate with a standard solution of potassium permanganate for the iron content and calculate as $FeSO_4$. Dissolve the copper remaining on the filter and in the beaker with a little dilute nitric acid, wash the filter with water into the beaker, add ammonia in excess, and titrate with standard potassium cyanide solution; calculate as $CuSO_4$.

Silver Present as Sulphate.—Take 0.2 A. T. and treat with hot water as above. If the copper in the solution amounts to more than 1 to 2% of the matte, use the following method. To the solution of the silver sulphate add NaCl solution of about normal strength to precipitate the silver as chloride, carefully avoiding an excess of the NaCl. Lead acetate and a few drops of sulphuric acid are added. Let stand for six hours, filter, scorify the precipitate, and cupel for silver. Calculate as $AgSO_4$. If the copper present is less than 1% the silver in the hot water solution may be determined by Volhards method. For total silver the scorification assay may be used.

RE-SULPHATIZING THE METALLIC SILVER

Metallic silver resulting from the decomposition of the silver sulphate by excessive heating, exists in a finely divided condition in the over-roasted matte. It may be re-sulphatized by the sulphuric anhydride evolved from decomposing ferrous sulphate. No doubt other decomposable sulphates would accomplish similar results, providing their dissociation occurred at a sufficiently high temperature. The sulphates of sodium and potassium, as such, do not decompose readily by heat alone. Ferrous sulphate is a convenient compound to employ, not only because of its comparative cheapness, but because it begins to decompose at a temperature far below that to which silver sulphate can be safely heated. Though it begins to decompose below 600° C., the decomposition is not rapid till 650 to 750° C. is reached. If a mixture of over-roasted matte and ferrous sulphate be rapidly heated to a temperature near 700° C.—that at which the basic sulphate dissociates—there will be little cupric sulphate formed, but the sulphuric anhydride fumes will act directly on the metallic silver. There will therefore be no necessity of raising the temperature or prolonging the time of the roast to decompose the sulphates of copper, thereby again running the risk of decomposing the silver sulphate.

Mix a portion of the over-roasted material with 5% of its weight of dehydrated ferrous sulphate and heat in the roasting dish in the furnace for 20 to 30 minutes. Allow the temperature to rise slowly above 590° C. and finally raise to 750° C. Take a sample and determine the silver present as sulphate and note the percentage of silver re-sulphatized by the roast.

In practice the silver is recovered by leaching the matte with hot water and precipitating the silver from this solution by passing over copper strips. The copper goes into solution as sulphate which is recovered by crystallization or by passing over scrap iron.

Report.—Plot all the data obtained in a series of curves, with time for ordinates, and temperatures, percentages of silver, copper and iron as sulphates, as insoluble compounds, and loss of silver as ordinates.

LIXIVIATION OF COPPER ORES

Principles.—The extraction of the copper in the ore by lixiviation is effected by the use of copper solvents such as sulphuric, hydrochloric, and sulphurous (H_2OSO_2) acids, and by air oxidation (Rio Tinto method). The ores suitable for direct treatment are the oxidized varieties that do not contain prohibitive amounts of the acid-consuming carbonates of lime, magnesia, alumina, and iron oxides or carbonates. Sulphide ores after being roasted with or without salt, will yield a satisfactory extraction with plain water leaching followed by a weak acid wash. The method might be successfully combined with the ammonia-cyanide process by removing the excess of copper over one per cent before the application of the ammonia cyanide.

Tests.—Take a sample of the ore and assay for gold, silver, and copper. Make a mineralogical examination for acid-consuming carbonates and to determine in what form the copper exists. If the copper is present as a sulphide, the ore must first be subjected to a roast as follows:

Take 500 grams of the ore, place in a cast-iron dish and roast in a muffle or oven furnace where the heat may be readily controlled. The object of the roast is to decompose the sulphides and form the maximum amounts of soluble copper salts. See 'Sulphatizing Roast,' page 86. A chloridizing roast may prove more effectual than a plain roast, as the precious metals are changed into more soluble forms, and as practised in the Henderson process* has proved very successful. If a chloridizing roast is used, proceed as described under the 'Chloridizing Roast of Silver Ores', page 66. When the roast is finished, cool, weigh, and assay.

Take 50 grams of ore, place in a 300-c.c. beaker with 150 c.c. of water and agitate at intervals for one-half hour. Decant through a filter and wash with 150 c.c. of water. Dry, and assay for gold, silver, and copper. The result will be the amount soluble in water alone.

Take 50 grams of ore, treat with water as above, return to the beaker, add 150 c.c. of 3 to 5% sulphuric or hydrochloric acid, decant through a filter, wash with 150 c.c. of water, dry and assay. A hot acid solution will give a higher and more rapid extraction but may result in an excessive consumption of the acid. The metal extracted by water washes subtracted from the above results will give amounts further soluble in acid due to the presence of basic compounds and oxides.

Consumption of Acid.—Take 20 grams of ore, wash thoroughly with water and introduce into a wide mouthed bottle with 40 c.c. of standard acid solution; agitate for 30 minutes and then filter off 20 c.c. of the acid. Titrate with a standard alkali. Calculate the consumption of 100% acid per ton of ore.

Report the extraction of gold, silver, and copper.

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*See Austin's 'Metallurgy of the Common Metals,' 2nd Ed., page 356.

Compare with net returns by shipping direct to smelter.

ELECTROLYTIC PROCESSES

GENERAL STATEMENT

The quantities of metal transported by a certain current in a certain time are proportional to the atomic weights of the metal divided by the valency in which it exists in the solution. This is known as 'Faraday's law.' Solutions containing the salts of the metal and generally with free acid present are almost entirely used in commercial electrolytic refining. Only those metals that do not dissolve from the anode with the evolution of hydrogen have been successfully refined up to the present. When a metal, such as sodium or aluminum, cannot be successfully treated in a liquid, a fused electrolyte is used. The deposition of pure metal in electrolytic refining depends upon the fact that each metal has its own electro-motive force of solution.* Metals lower in the scale than the principal metal present are eliminated as metal particles in the anode slime, and metals higher in the series are eliminated as salts dissolved in the solution or precipitated from it.

The transportation of a pure metal from one pure anode to another in the same physical condition, requires little power. In commercial work time is an important factor, as the capacity of the plant varies inversely with the speed of working, hence a bal-

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*The difference in voltage existing between a metal, and a solution containing the same metal, in which it is immersed.

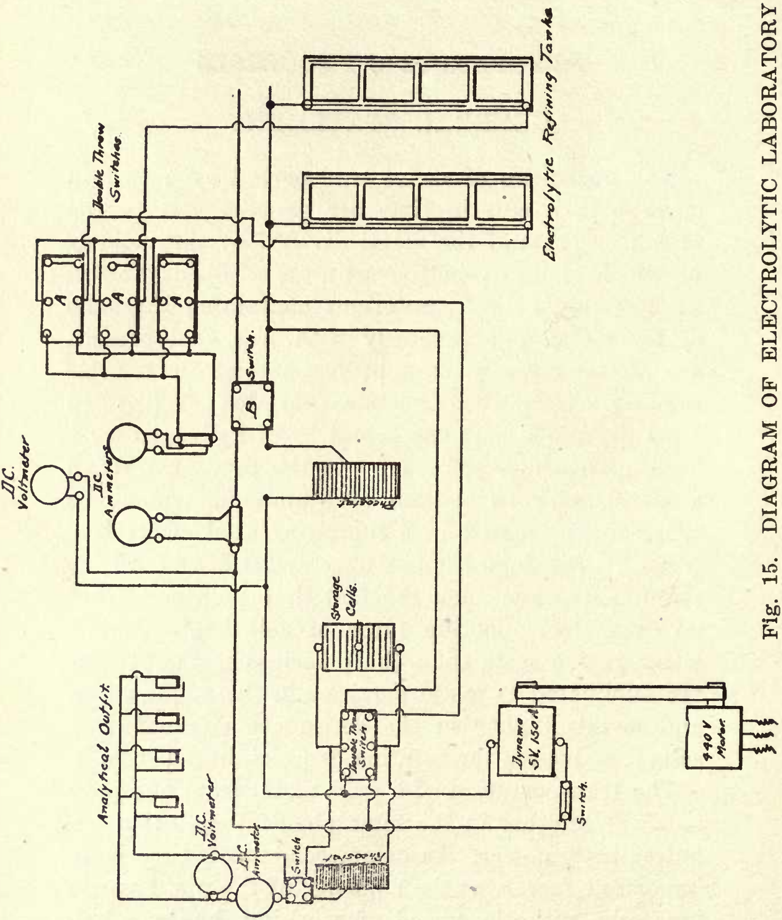

Fig. 15. DIAGRAM OF ELECTROLYTIC LABORATORY

ance must be found between an increased cost for a greater output against lower costs for power for a decreased output, per vat, in the same time. Since power is such an item of expense it is one of the first considerations to find an electrolyte of as high electric conductivity as possible.

The following electromotive forces of solution are practically correct for fluosilicate solutions, but may vary a few hundredths of a volt for different strengths of electrolyte and also for different solutions:

	Volts.
Zinc	+ 0.52
Cadmium	+ 0.16
Iron	+ 0.09
Lead	— 0.01
Tin	— 0.01
Arsenic	— 0.40
Antimony	— 0.44
Bismuth	— 0.48
Copper	— 0.52
Silver	— 0.97
Mercury	— 0.98

ELECTROLYTIC REFINING OF COPPER

Principles.—The process depends upon the fact that when copper-bearing alloy, cast into slabs, is used as the anode of an electric current, an acid solution of copper sulphate as the electrolyte, refined copper as the cathode, and a current of suitable density is employed, the copper will be dissolved from the anode and deposited at the cathode, while the gold and silver will remain at the anode and fall to the bottom of the tank in the form of mud. At the anode SO_4 is separated and a corresponding amount of copper is dissolved; through this solution of the copper the electric current is reinforced by an amount of energy that, to a great extent, neutralizes the back electromotive force. In the process impurities have various effects. Gold, silver, and platinum are precipitated as metallic particles in the slime. Lead becomes a sulphate. Arsenic, bismuth, and antimony go into solution and are precipitated as basic sulphates, but as the electrolyte becomes neutral or foul the arsenic and bismuth are partly carried over and precipitated at the cathode. Tin dissolves and is precipitated as a basic sulphate. However, it is not harmful, but improves the cathode deposit and diminishes the tension of the bath. Iron, zinc, nickel, and cobalt dissolve and impoverish the bath as regards both copper and free acid. The neutralization of the solution has many prejudicial effects upon the electrolytic process. The solution becomes a poorer conductor and foreign metals are deposited at the cathode along with cuprous oxide.

In order to produce good copper the solution must circulate as actively as possible, because copper deposited at the cathode will be purer and more malleable. If this solution is not in circulation it becomes richer in copper at the anode and poorer at the cathode; the poorer solution ascends in the bath, thus introducing a higher resistance in the circuit. This renders it possible for other metals or hydrogen to be separated along with the copper. As regards the relation between the strength and tension of current, the same amount of copper can be obtained with a given power, whether the voltage is high or low, as long as the current density is the same; i. e., the number of amperes per square foot of anode surface. One ampere will deposit approximately one ounce of copper per 24 hours.

The electrolyte is purified by aerating; boiling with metastannic acid; filtration through cuprous oxide; by replacing with new electrolyte and working the old for copper and sulphate of copper; or depositing the arsenic by taking a portion of the electrolyte continuously, sending through a tank with insoluble lead anodes and copper cathodes, where the arsenic and excess copper are deposited together by a high density current. Experience shows that the liquid most suitable for electrolysis contains per gallon 0.5 pound of concentrated sulphuric acid (H_2SO_4) and 5 pounds of bluestone containing 0.38 pounds of copper. A convenient way of making up the electrolyte is to take by weight: water, 75 parts; bluestone ($CuSO_4 + 5H_2O$), 16 to 18%. After the bluestone is dissolved, add sulphuric acid to make up to 16 to 18° Beaumé (sp. gr. 1.12 to 1.15). A small amount of salt is sometimes added to precipitate antimony in the electrolyte as oxychloride and also to act as a guard and prevent the silver from going into solution.

While there are many so-called processes of electrolytic refining, all are ultimately reduced to two systems, and according as the electrodes in such cases are in multiple arc or in series they are known as the **multiple** and the **series** systems, respectively. In the multiple system an arrangement is contemplated where the electrodes are in parallel arc, the tanks being in series; while in the series system there is intended an arrangement where the electrodes are in series. The tanks may be either in series or multiple series. From a consideration of electrical principles it is obvious that the voltage required per tank will depend, neglecting the constant of the solution, upon the area of the plates, their spacing, and the number in series. The multiple system is applicable to all grades of copper, it permits the use of both high and low current-density in lead-lined baths, and it can be carried out with little loss of current as compared with the series system.

Procedure.—Cast an anode of copper which contains impurities usually found in anode copper, such as O_2, S, As, Sb, Fe, Pb, Au, and Ag. Before pouring, take a sample of the molten copper and granulate by pouring slowly from the ladle into a pail of water; reserve this sample and assay for copper and impurities. Prepare the anode mold by giving it a wash of clay, lampblack, or graphite and heating on top of the furnace. The anode should be cast approximately one-fourth of an inch in thickness. Prepare electrolyte to fill one tank and provide for circulation. In case sufficient electrolyte is left from a previous experiment, assay for $CuSO_4$ and H_2SO_4 and bring up to the required strength. Carefully weigh the anode and place in position in the tank. Care should be taken to have all contact surfaces clean and bright. They should be flat so as to give as little resistance as possible. Weigh the cathodes and place in position, using one more cathode than the number of anodes in the tank.

When the anodes and cathodes are in position and the circulation of the electrolyte started, throw on the current from the dynamo, using 15 amperes per square foot of anode surface exposed. Analyze the electrolyte each day for $CuSO_4$ and H_2SO_4 and keep up to the required standard by the addition of bluestone or sulphuric acid as required. Observe all precautions mentioned in the notes so as to obtain a firm, pure deposit of copper on the cathode. Keep a log showing each day the number of ampere hours run, the number of anodes in the circuit and their arrangement, analysis of the electrolyte, drop in voltage between each anode and cathode, appearances of the deposit, and general remarks. The drop in voltage between the plates is measured by a forked stick provided with contact wires leading to a volt-meter. This precaution is taken to detect any short-circuit and observe any unduly high resistance from poor contacts or from coatings of slime on the anodes. The latter may be removed by scrubbing with a stiff brush.

At some period during the run prepare at least two stripping plates as follows: Take a regular anode made of copper sheeting from $1/8$ to $1/16$ in. thick, make a coating of paraffine $1/8$ in. wide around the edges, covering the remainder of the surface with graphite. Hang in the circuit in place of one of the regular cathodes and allow the copper to deposit for 24 hours. The paraffine on the edges of the plate and the graphite coating allow the resulting deposit to be easily 'stripped' off in a sheet which is used as a cathode in succeeding runs.

At the end of the run the anodes and cathodes are removed, washed, dried, and carefully weighed.

Save the slime washed from the anode and add to that taken from the bottom of the tank. The loss in weight of the anodes should approximately equal the increase in weight of the cathodes (do not neglect copper deposited on stripping plates) plus the weight of the slime. Compute the efficiency of the run. Knowing the number of ampere hours run, the theoretical deposition can then be determined as follows:

$$\frac{\text{actual deposition}}{\text{theoretical deposition}} \times 100 = \text{per cent efficiency}$$

Make an analysis of the cathode copper for arsenic and other impurities. The percentages of these present will indicate the efficiency of the run. Carefully siphon off the electrolyte from the tank so as not to disturb the sediment. Recover the slime by draining the contents of the bottom of the tank through a large filter. Wash free from acid and salts, dry, and refine. Recover the gold and silver from the bullion by cupellation and parting.

ELECTROLYTIC REFINING OF LEAD—BETTS PROCESS

Principles.—The process is based on the fact that lead is easily soluble in an acid solution of lead fluosilicate, which possesses both stability under electrolysis and high conductivity, and from which exceptionally pure lead may be deposited from impure anodes. The cost of the operation is comparatively low. With such a solution there is no polarization from the formation of lead peroxide at the anode, no evaporation of the constituents, except water, and no danger in handling. In order to obtain a firm deposit of lead on the cathode it has been found necessary to make the solution reducing by the addition of gelatine or pyrogallol. During the electrolysis the SiF_6 ions travel toward the anodes and there combine with the lead. The lead and hydrogen travel in the opposite direction and out of the slime. There are comparatively few lead ions present, so that the solution in the neighborhood of the anodes must increase in concentration and tend to become neutral. This causes an E. M. F. of polarization to act against the E. M. F. of the dynamo, amounting to approximately 0.2 of a volt for each tank. The greater effect comes from the neutral solution with which the slime is saturated. There is, consequently, an advantage in working with rather thin anodes when the bullion is impure enough to leave the slime sticking to the plates. The fact that the slime sticks to the anode is compensated by the increased ease of handling the slime, by removing with the anodes to special cleaning and washing tanks.

Lead stands higher in the scale of electromotive forces of solution than any of the impurities that it contains in appreciable amounts, so that the electrolyte does not need to be changed or purified, as in copper refining, when treating the ordinary grades of bullion. The electro-chemical equivalent of lead is also high; one ampere depositing 3.88 grams of lead per hour, or 3¼ as much lead as copper in the ordinary copper-refining solution.

The metallic elements that enter into consideration as possible constituents of the electrolyte are the elements usually present in the lead bullion, those that may be in the fluosilicic acid as impurities at the start, and iron from any exposed binding of the tanks. Arsenic, antimony, seleninum, and tellurium are easily precipitated by lead, and consequently if they get into solution they will be thrown down by the lead electrodes; probably by the cathodes, for the anodes are usually covered with slime. This would prevent their reducing, for instance, much antimony, although the antimony of the slime would throw out such an easily precipitated metal as silver. Zinc, iron, and nickel, if they find their way into the solution, stay there; for they are not precipitated by the lead electrodes, nor can they be in any way thrown out on the cathodes by the electric current so long as there is a fair amount of lead in solution, as is always true. It is doubtful whether the iron goes into solution at all, as the slime usually assays from ½ to 2% of this metal. Tin occupies practically the same position in the scale of electromotive forces that lead does, consequently a mixture of lead and tin will behave practically as one metal. Alloys of copper and lead give a slime practically free from lead. Alloys of 40% copper and 60 lead, can be successfully treated. Silver retains very little lead in the slime. Bismuth holds back $^1/_6$ and antimony $^1/_4$ to $^1/_5$ its weight in lead.

The electrolyte is prepared by saturating 35% hydrofluoric acid with quartz, adding the required amount of lead as carbonate and gelatine in the form of glue. The amount of gelatine consumed under good working conditions is only ½ to ¾ pound to the ton of lead deposited. Gelatine in the form of glue is always used and is cheaper. The better grade of glue should be used, as the cheaper varieties make a disagreeable smell in the tank room. The lead deposit, forming in lead-fluosilicate fluosilicic acid solution, containing 0.1% gelatine and 5% or more of lead, is smooth and solid, and pieces cut from the cathode show a specific gravity of 11.35 to 11.40 (the specific gravity of cast lead). With a little more lead and the average current density employed in commercial operations, about 15 amperes per square foot, the resulting cathodes, after reaching a considerable thickness, are smoother. The smoothness and purity of the deposited lead are proportional. Most of the impurities seem to be introduced mechanically through the attachment of floating particles of slime to the irregularities of the cathode.

The anodes and cathodes are spaced from 1½ to 2 inches. A current strength of 10 to 25 amperes has been used, but 14 amperes gives the best results as regards the economy of working and the physical and chemical properties of the refined metal produced.

Procedure.—The method in treating lead is practically the same as in the electrolytic refining of copper. The fluosilicic acid is prepared by putting hydrofluoric acid of 15 to 20% strength in a lead pan and adding an excess of powdered quartz or, better, calcined flint which dissolves more readily. Heat the pan, but not so much as to cause the acid to boil, until the solution is saturated with silica, or until the pungent odor of HF has ceased. To make the lead solution, add the required amount of white lead, which ordinarily contains 80% of metallic lead. Gelatine or glue is added to the electrolyte in the form of a strong hot solution in water.

The electrolyte is made up so as to contain:

10 to 12% H_2SiF_6
5 " 9% Pb
0.1 " 0.2% gelatine
Sp. gr. 1.16 (approximately)
Temperature during electrolysis 34° C.

To determine acidity of electrolyte, add alcoholic potassium acetate solution. Filter off K_2SiF_6, wash with diluted alcohol, add filter and precipitate to distilled water in a beaker, heat to boiling and titrate with NaOH,* using rosolic acid preferably, but also phenolphtalein as an indicator. To determine lead, add H_2SO_4, filter, and determine by molybdate method. Other determinations are seldom made. For more detailed information, and for methods of slime treatment see 'Lead Refining by Electrolysis' by Betts.

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*1 gram of NaOH = 0.9 gram H_2SiF_6.

ZINC SMELTING

General Process.—The zinc-bearing minerals seldom occur pure and are usually associated with other metaliferous, silicious or earthy gangue minerals. Chemically combined impurities such as iron are also almost invariably present. The mechanically combined minerals may be separated to a greater or less extent by methods of ore dressing, but for the elimination of chemically combined impurities a metallurgical treatment is required.

The carbonates and silicates of zinc are usually calcined preliminary to retorting, in order to drive off the carbon-dioxide and water. In roasting blende preliminary to the distillation of the zinc, the aim is to convert the sulphide as completely as possible into the oxide, since every part of sulphur remaining behind in the roasted ore means approximately two parts of zinc held in the retort after distillation. The accomplishment of this object requires fine communition of the zinc blende, generally not to exceed 2 mm. size (6-mesh), high temperature, and slow, careful roasting with frequent stirring. It is never economical to effect a complete elimination of the sulphur, which is impossible with some ores. In good practice the amount in combination with zinc, iron, and lead is reduced to one per cent and frequently lower. Ores containing lime and magnesia may retain much more owing to the formation of calcium and magnesium sulphates. These last two sulphates are formed by the action of sulphuric anhydride on their respective oxides. Calcium sulphate is not decomposed at the temperature attained in the roasting furnace, but magnesium sulphate is partly decomposed in the last stages of the process.

Chemical Reactions.—Starting at a dull red heat,

$$ZnS + 3O = ZnO + SO_2$$

$$-41880 = +85430 + 71000 = +114550 \text{ calories}$$

This reaction is exothermic and develops sufficient heat to proceed independently until the sulphur is burned down to 5 to 8% according to the conditions.

At a bright red heat, $SO_2 + O = SO_3$, by contact (catalysis) with the glowing ore and furnace walls, and $ZnO + SO_3 = ZnSO_4$.

At a cherry-red heat the zinc sulphate breaks up into basic sulphate ($3ZnO.ZnOSO_4$) and SO_3. The SO_3 is in turn broken up more or less into SO_2 and O. The basic zinc sulphate is not broken up entirely until it has been exposed to a bright red heat for a considerable time where there is danger of the loss of zinc by volatilization, and the sintering of the charge when lead, iron, or manganese are present. An experienced roaster can judge by the volume and cessation of the fumes over the ore the different stages and time of completion of the roast. The presence of sulphates, which is an important factor, can only be determined by analysis.

Distillation.—The zinc in the ore, having been converted into oxide by calcinating or roasting, the recovery of the metal is accomplished by virtue of the fact that the oxide is reducible, when heated to a high temperature, by carbon or carbon monoxide or both, according to the reaction:

$$\mathrm{Zn} + \mathrm{C} = \mathrm{Zn} + \mathrm{CO}$$
$$-86000 = +29000 = -57000 \text{ calories}$$

This reaction takes place, according to different authorities, from 800° C. to 1075° C., but practically a temperature of 1300° C. is required. The metal is reduced in the form of vapor which is subsequently liquefied by cooling to about 415 to 550° C. All the zinc vapor does not condense as a liquid but 5 to 10% of it passes directly into the solid state in the form of a bluish powder called zinc fume. The presence of silica, alumina, lime, magnesium, iron, manganese, and lead is detrimental when they are in proportion to form fusible slags which corrode the retort and coat over the particles of ZnO, thus causing a loss of zinc in the residues. When cadmium, arsenic, and antimony are present they are partly condensed with the zinc.

Procedure in Testing.—Take a 500-gram sample of the ore and crush to 8 mesh. Sample and assay the original ore for zinc, sulphur, and impurities. Place the 500 grams of ore in a large roasting dish and put in place on the hearth of the gas-oven furnace. Observing carefully all the precautions previously given, roast down to the lowest practical limit of sulphur. At the end of the operation weigh the ore and take a sample and assay for sulphur and zinc. Mix the roast ore with 50% of its weight of crushed coal and charge into a retort, which has been slowly brought up to a white heat in the gas forge. Put the condenser in place and keep at a good white heat until the fumes of zinc cease to appear at the mouth of the condenser. Allow to cool down slowly, remove the residue from the retort, weigh, sample, and assay for zinc. Collect the spelter and zinc fume and weigh separately and assay for zinc. Note the condition of the interior of the retort.

Report.—Efficiency of the roast, that is, loss of zinc and sulphur, percentage of recovery of zinc by retorting, cost data, should be tabulated for ready reference.

FURNACE TEST

General.—The object of this test is to determine as nearly as possible the heat efficiency of a furnace as well as the losses. The starting point will evidently be the determination of the calorific power of the fuel used. This will furnish means for a comparison of the quantities of heat lost and utilized in the furnace. The next point is to construct a heat balance sheet of the furnace in question for the purpose of placing in a clear and concise manner the ratio of heat used to the total available heat, also to the heat losses. The value of a furnace can be determined with a certainty, and if the efficiency is below a required standard the cause of such a deficiency may be deducted from the test.

The determination of the heat balance requires the following: (1) heat utilized; (2) heat carried away by the waste products; (3) heat lost by conduction and radiation. The sum of these three quantities must be equal to the total available heat, the calorific power of the fuel. It also follows that when any two of the quantities are known, the third may be determined by difference.

The experimental data required are the following: (1) calorific power of the fuel; (2) elementary analysis of the fuel and ash; (3) analysis of the gaseous products; (4) temperature determinations at various points in the furnace.

Before proceeding with the test, determine the specific object of the proposed trial, whether it is to ascertain the efficiency of the furnace or its defects, the economy of a particular kind of fuel, or the effects of change of design, proportion or operation, and prepare for the trial accordingly. Ascertain the dimensions of the grates and important parts. Clean the furnace thoroughly and stop any air leaks in the setting. Before the trial see that the furnace is heated to its usual working temperature. If it has been laid off and become cold, it should be worked before the trial until the walls are well heated.

The standard method of starting and stopping a test is with clean grate-bars and ash-pit, but the alternate method, while not so accurate, is more convenient. The fire is burned low and well cleaned, and all the ashes removed. Note the amount of coal left on the grate, as nearly as can be estimated by the eye, and take the temperature of the flue gases. At the end of the test the fires should be burned low and cleaned so as to have the conditions as nearly as possible the same as at the start. Fair samples of the coal and ash are to be taken and enclosed in air-tight cans. The total weight of both coal and ash are to be accurately determined. A sample of the flue gas is drawn off continuously during the experiment by means of an aspirator. The temperatures of the flue gases are taken by means of a thermo-electric pyrometer. The hot junction is placed near the gas-sampling tube and the lead wires run to a convenient place for the galvanometer. The stack suction is determined by means of a U tube, or preferably with a draft gauge graduated to read to $^1/_{100}$

of an inch water column. A quarter-inch iron pipe is inserted in the flue and connected to the draft gauge by means of a suitable length of rubber tubing.

Resume.—Put all the apparatus in place for taking the necessary reading.

Introduce into a hole in the stack the gas-sampling tube, pyrometer junction, and the draught-gauge tube as shown in Fig. 16. Place the instruments where they will not be affected by the heat, not liable to damage and can be easily read.

Clean the fire and ash-pit. Note the thickness of fire and condition of the bed, also temperature of flue gas.

Weigh out enough coal for the first firing, as nearly as can be judged; take a sample of the coal. Take initial readings of the galvanometer and draft gauge and start the aspirator to taking the gas sample.

Put coal on the dead plate and a thin layer over the bed of the fire. Note the time of commencing and of firing.

Take necessary readings of draft gauge and pyrometer every five minutes.

Fire up, when necessary, to keep the muffles at a temperature required for good assay work.

Run the gas aspirator so as to fill two sample bulbs during the hours run. Let the gas be drawn through the sample tube for ten minutes before removing it, so as to have the flue gas thoroughly replace the air in the tube.

At the end of the test take a sample of the ash and determine its sensible heat by dumping into a

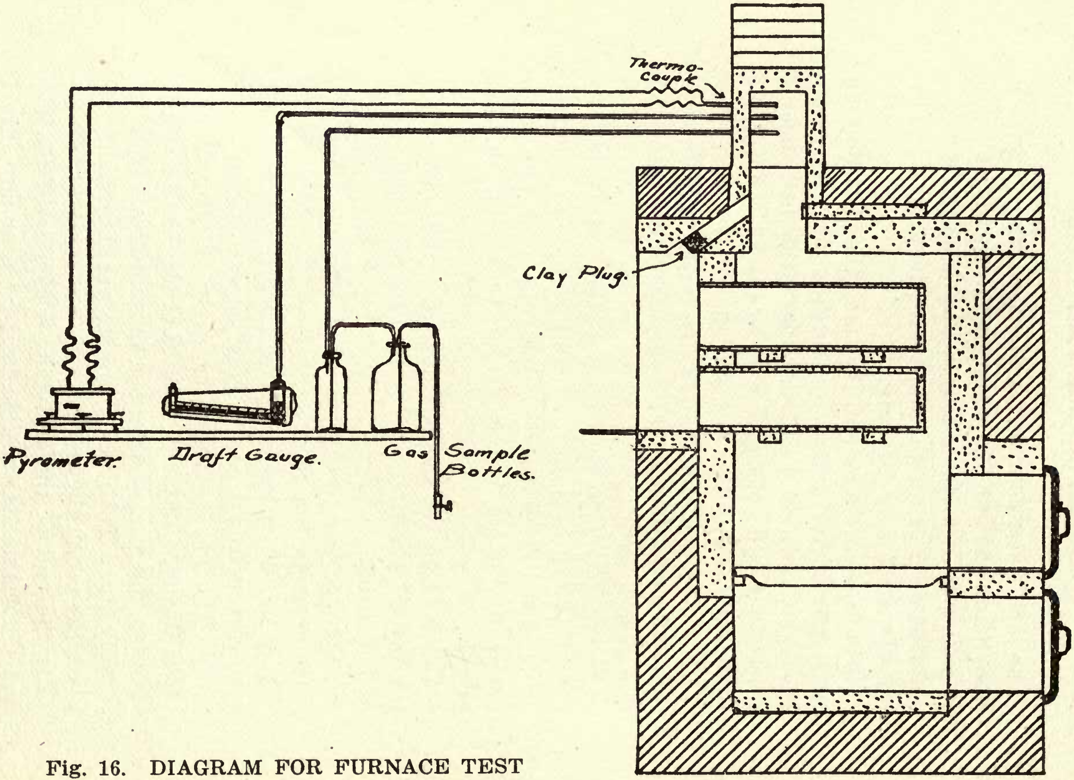

Fig. 16. DIAGRAM FOR FURNACE TEST

pail containing a known amount of water and weight.

Make an analysis of the two gas samples with the Orsat gas apparatus, determining CO_2, CO, O, and N (by difference).

Make analyses of coal and ash, and determine their calorific power.

Report

Calorific value 1 lb. coal =.... lb. cal 100%.

Distribution	Lb.Coal	%
Heat used usefully....................		
Calorific value ash from 1 lb. coal......		
Loss due to moisture in coal*..........		
Loss due to sensible heat carried away in dry chimney gases..............		
Loss due to incomplete combustion of carbon, etc., in flue gases..........		
Loss due to moisture in the air, radiation and unaccounted for, to make up to 100%		
Totals		

*All calculations computed on the basis of 1 lb coal.

Pounds of coal burned per square foot of grate area, per hour = ...

METHODS OF CALCULATION FOR HEAT-BALANCE SHEET

Heat Used Usefully.—The calculations under this head will vary with the nature of the work done in the furnace and is not taken into account in the test of assay furnaces. In roasting, the heat equations of the various constituents of the ore must be included. In fusion, the heat required to fuse the charge must be taken into account.

Loss of Heat Due to Moisture in the Coal.—This may be determined by the following formula:

P = per cent moisture in the coal.
t = temperature of the room.
T = temperature of the flue gas.
C = thermal capacity of water vapor at temperature t.
C_1 = thermal capacity of water vapor at temperature T.

$$\frac{P}{100} \times ((100 - t) + 518 + (C_1 - C)) = \text{calories.}$$

Where 100 = boiling point of water.

518 = latent heat of vaporization of water.

The values for C_1 and C are read off the curve; see Fig. 17.

Loss Due to Heat Carried Away in Dry Chimney Gases.—This equals the sum of the thermal capacities of the constituents of the flue gas.

The thermal capacity of each gas is,

$$\text{weight gas} \times (C_1 - C) = Q$$

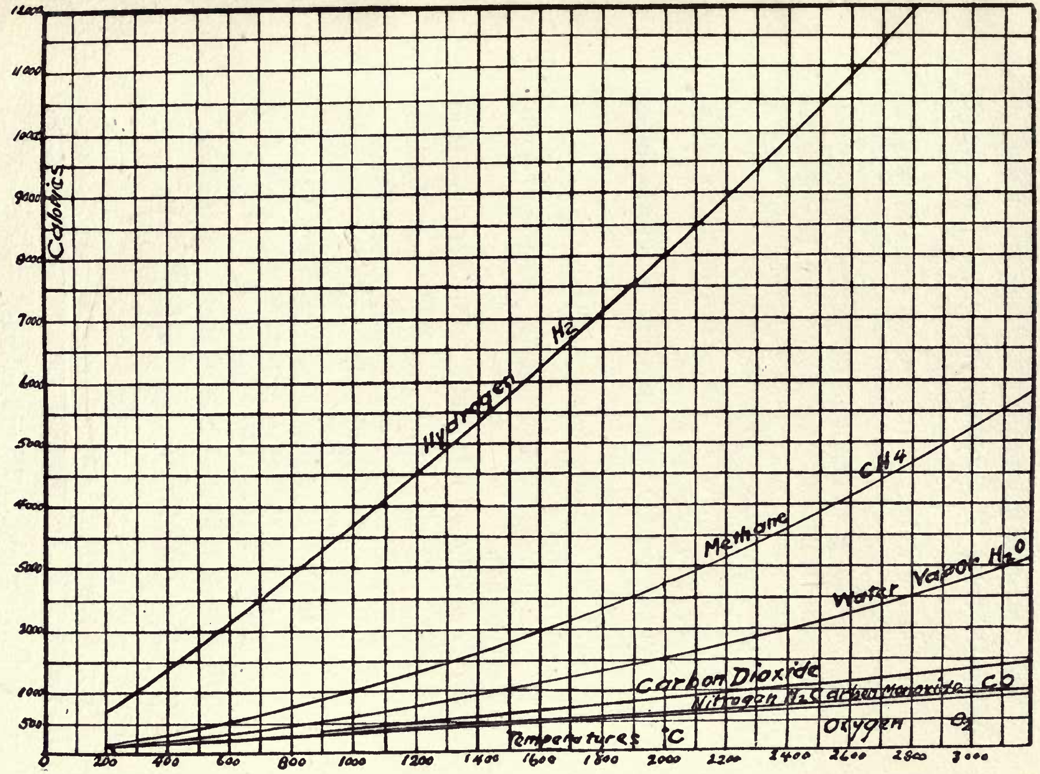

Fig. 17. DIAGRAM GIVING THE THERMAL CAPACITIES OF GASES PER POUND FOR ANY TEMPERATURE BETWEEN 0° AND 3200°C. (Dissociation is here neglected).

The combined thermal capacities would be,

$$\text{Wt. } CO_2 \times Q + \text{wt. } CO \times Q + \text{wt. } O \times Q + \text{wt. } N \times Q + \text{etc.} = \text{calories.}$$

To find the weight of the several gases proceed as follows:

Suppose the gas to contain CO_2, CO, O, and N.

(1) The molecular volume of any gas is 22.38 liters = M, or the molecular weight of any gas in grams has a volume of 22.38 liters = M.

(2) Let WCO_2, WCO, etc., stand for the molecular weights of the various substances.

(3) Let CO_2, CO, etc., stand for the percentages by volume.

In one unit volume of gas we have,

(4) $\frac{CO_2}{100} + \frac{CO}{100}$ = volume of $CO_2 + CO$ in 1 unit volume of gas.

(5) According to (1) and (2) if $WCO_2 = M$, 1 unit volume of CO_2 will weigh $\frac{WCO_2}{M}$ and similarly 1 unit volume of CO will weigh $\frac{WCO.}{M}$

But in (4) we have the volume of $CO_2 + CO$.

(6) $\frac{CO_2}{100} \times \frac{WCO_2}{M} + \frac{CO}{100} \times \frac{WCO}{M}$ = weight $CO_2 +$ CO in 1 unit volume.

(7) But CO_2 is $^{12}/_{44}$ carbon $\left(\frac{(WC)}{WCO_2}\right)$ and CO is $^{12}/_{28}$ carbon.

$$\frac{CO_2}{100} \times \frac{WCO_2}{M} \times \frac{WC}{WCO_2} + \frac{CO}{100} \times \frac{WCO}{M} \times \frac{WC}{WCO}$$

= weight carbon in 1 unit volume.

Combining and substituting values,

(8) $\frac{12}{2238}(CO_2 + CO)$ = weight carbon in 1 unit volume.

The carbon in the fuel — the carbon in the ash = carbon burned to flue gas.

From (8) we have the weight of carbon in 1 volume of flue gas.

From 1 unit weight of carbon burned to flue gas containing CO_2, CO, O, N, we will get,

$$(9)\qquad \frac{1}{\frac{12(CO_2 + CO)}{2238}} = \text{unit volumes of flue gas.}$$

According to (1) — (4) and (5),

$$(10)\qquad \left.\begin{array}{l} \text{Weight } CO_2 \text{ in 1 volume gas} \\ \qquad\qquad CO \\ \qquad\qquad O \\ \qquad\qquad N \end{array}\right\} = \frac{\%\text{ gas} \times \text{W gas}}{100M}$$

And from (9) the unit volumes of gas.

Then,

$$(11)\qquad \frac{CO_2 + WCO_2}{100M} \times \frac{1}{\frac{12(CO_2 + CO)}{2238}} = \begin{array}{l}\text{Wt. } CO_2 \text{ from 1} \\ \text{unit of carbon} \\ \text{burned.}\end{array}$$

Substituting values and combining.

$$(12)\qquad \frac{11CO_2}{3(CO_2 + CO)} = \begin{array}{l}\text{weight of } CO_2 \text{ from 1 unit of car-} \\ \text{bon burned.}\end{array}$$

In a like manner,

$$(13)\qquad \frac{7CO}{3(CO_2 + CO)} = \begin{array}{l}\text{weight of CO from 1 unit of car-} \\ \text{bon burned.}\end{array}$$

(14) $\frac{7N}{3(CO_2 + CO)} =$ weight of O from 1 unit of carbon burned.

(15) $\frac{8O}{3(CO_2 + CO)} =$ weight of N from 1 unit of carbon burned.

Now having the weight of the constituent gases per unit weight of carbon burned, the same per pound of coal burned may be calculated.

a = weight of coal burned.
b = weight of ash obtained.
c = percentages of carbon in coal.
d = " " " " ash.

(16) $\frac{a \times \frac{c}{100} = b \times \frac{d}{100}}{a} \times 100 =$ per cent available carbon in the coal.

Or, $\frac{ac - bd}{a} =$ per cent available carbon in the coal.

Therefore the weight of any constituent gas × per cent available carbon in the coal ÷ 100 = weight of that gas per unit weight of coal burned.

(17) Example. $\frac{8O}{3(CO_2 + CO)} \times \frac{ac - bd}{100a} =$ weight of O per pound of coal.

As previously given the heat carried away by a gas per unit of weight of coal.

(18) Wt. gas per lb. coal $(C_1 - C) =$ calories.

$C_1 =$ thermal capacity of gas at stack temperature.
$C =$ " " " " " room "

The value for C and C_1 are taken from the curve, Fig. 17 or may be computed for any temperature as follows:

(19) $Q = a \times t + b \times 10 - 6t^2.$

Where a and b are factors variable with different gases.

	a	b
Oxygen, O	0.213	19
Nitrogen, N; carbon monoxide, CO	0.234	21
Hydrogen, H	0.340	300
Water vapor, H_2O	0.447	162
Carbon dioxide, CO_2	0.193	84
Methane, CH_4	0.608	374

Total heat carried away in dry chimney gases,

Let w = weight of gas per pound of coal (17)

Q = thermal capacity.

(20) $wCO_2Q + wCOQ + wOQ + wNQ =$ calories.

(d) Loss due to incomplete combustion of carbon in the flue gases, represented by the available calories of the CO.

As given under Ex. (17)

(21) $$\frac{7CO}{3(CO_2 + CO)} \times \frac{ac - bd}{100\,a} = \text{weight of CO per pound of coal.}$$

Calorific value of CO = 2436.

(22) Weight of CO per pound of coal × 2436 = calories available in CO.

(e) Heat value of ash. See 'Calorimetry.' (P. 123.)

Heat value of 1 pound of ash times weight of ash obtained divided by weight of coal burned equals the heat value of ash per pound of coal.

(f) Sensible heat of ash. See 'Furnace Test, Resumé.' (P. 113.)

In the above calculations ethylene, C_2H, methane, CH_4, etc., are not taken into account. If separate determinations of these gases are obtained the results may be used to make the balance sheet more complete and accurate, by making corrections for these gases along lines already given.

For instance (8) would be changed to,

(23) $\dfrac{1}{\dfrac{12(CO_2 + CO + \frac{C_2H_4}{2} + CH_4)}{2238}} =$ unit volumes of flue gas per unit weight of carbon burned.

Letting $3(CO_2 + CO + \dfrac{C_2H_4}{2} + CH_4) = a.$

(24) $\dfrac{11CO_2}{a} =$ weight CO_2 per unit weight carbon burned.

(25) $\dfrac{7CO}{a} =$ " CO " " " " "

(26) $\dfrac{7N}{a} =$ " N " " " " "

(27) $\dfrac{8O}{a} =$ " O " " " " "

(28) $\dfrac{7C_2H_4}{a} =$ " C_2H_4 " " " " "

(29) $\dfrac{4CH_4}{a} =$ " CH_4 " " " " "

(30) $\dfrac{H}{2a} =$ " H " " " " "

The above values calculated in terms of a unit weight of coal burned as in (17) would then be substituted in (20).

$QwCO_2 + QwCO + QwO + QwN + QwC_2H_4$, etc. $=$ calories carried away in dry gases.

APPROXIMATE DETERMINATION OF THE CALORIFIC POWER OF A FUEL

Few industrial laboratories are equipped with calorimeters, and it is often desirable to find within 1% (approximately) the calorific value of a fuel. In this case the Malher and Goutal formulas are satisfactory and have stood the test of practice. The Goutal formula makes use of the data obtained from the ordinary approximate analysis; the Malher of the ultimate analysis.

PROXIMATE ANALYSIS

Moisture Determination.—The moisture is determined on a 2-gram sample heated for one hour in an air bath at 115°C. The constant weight should be verified by two weighings.

Volatile Matter.—In a crucible of 30 c.c. capacity is placed a 5-gram sample of the powdered fuel, the crucible is placed uncovered over a Berzelius burner. The flame is maintained so as to completely surround the crucible. After all illuminants have been entirely driven off, heating should be continued for three more minutes.

Ash.—Two grams of fuel are burned in a muffle until only a white residue remains, the heating, to be done very carefully and not too suddenly.

Fixed Carbon.—The fixed carbon is determined by difference between the sum of the other constituents and 100 per cent.

Example.—The analysis of an anthracite is as follows:

	Per cent.
Fixed carbon (C)	86.70
Volatile matter (V)	10.05
Ash	1.45
Moisture	1.80
	100.00

The percentage of volatile matter V′, calculated as if there were neither ash nor moisture is,

$$V' = \frac{V \times 100}{C + V} = \frac{10.05 \times 100}{86.70 + 10.05} = 10.04$$

Goutal Formula.—The data obtained by this analysis may be used in calculating the approximate calorific value of the fuel by the following formula:

$C =$ fixed carbon.

$V =$ volatile matter.

$a =$ a variable factor.

a is a function of the volatile matter in the fuel, allowance being made for moisture and ash.

Calorific value $= 82C + aV$.

For all fuels for volatile matter V′ below 40% the values from a are given in table following.

CALORIFIC POWER OF FUEL—GOUTAL FACTORS

V′%.	*a* Calories.	V′%.	*a* Calories.	V′%.	*a* Calories.
1 to 5	100	17	113	29	99
5	145	18	112	30	98
6	142	19	110	31	97
7	139	20	109	32	97
8	136	21	108	33	96
9	133	22	107	34	95
10	130	23	105	35	94

11	127	24	104	36	91
12	124	25	103	37	88
13	122	26	102	38	85
14	120	27	101	39	82
15	117	28	100	40	80
16	115				

From the table we find that V′ (10.04) corresponds to a value of *a* equal to 129 calories. Substituting in the formula we have

$$\text{Calorific power} = 82 \times 86.7 + 129 \times 10.05 = 8406 \text{ calories.}$$

By actual determination on a bomb calorimeter it was found to be 8404 calories. The results obtained with the formula are generally within 1% of the experimental determinations, and an error of 2% is quite exceptional and only observed with anthracites and lignites. For the more exact determinations of the calorific power of a fuel, a colorimeter is used. The use of these instruments is fully explained by booklets furnished with the outfit.

MALHER FORMULA

When an ultimate analysis is available the calorific power may be calculated as follows:

Q = calorific power.

$$= \frac{8140C + 34500H - 3000(O + N).}{100}$$

Substituting in the equation we finally get.

$Q = 111.40C + 375H - 3000$ in calories per kilogram of fuel.

This formula does not give accurate results in the case of cannel coal, lignite, peat, and wood.

PYROMETRY

General.—In the preceding chapters it has been shown that it is necessary in making certain tests to determine accurately and quickly the comparatively high temperatures of furnaces, flues, etc. The instrument selected for the work should be portable, accurate, easy to manipulate, and should allow the reading of temperatures of widely distant points. Of the many types the LeChatelier thermo-electric pyrometer fulfills best the above mentioned conditions. The thermo-electric instrument being small can be introduced into any part of the furnaces, and since a few seconds are sufficient for establishing an equilibrium of temperatures, it serves to measure temperatures close on to 1780°C., the melting point of platinum, without endangering the life of the wires. The LeChatelier pyrometer is thoroughly reliable, providing care is taken with the wires. However, in order to be successful with the instrument, its limitations should be borne in mind. These are due entirely to the nature of the metals forming the couple. Practically the same care that is to be taken with platinum ware, is required with the wires of the thermo-couple. Platinum is readily attacked by the vapors of volatile metals. Silver, zinc, antimony, and copper are especially injurious. In a reducing atmosphere silicates and phosphates are objectionable. In spite of the most thorough protection, platinum and its alloys suffer physical alterations, resulting in great brittleness by mere continuous heating to temperatures above 1000°C.

For measuring the E. M. F. of the thermo-electric couple, a galvonemeter reading to millivolts is used. Any two metals may be used for the thermo-electric couple, but the following are commonly employed.

1. Pure platinum and an alloy of 10% rhodium and 90% platinum (or 10% iridium and 90% platinum).

2. Pure silver (or copper) and alloy of 25% nickel and 75% copper (Constantan).

3. Pure silver and an alloy of 25% platinum and 75% silver.

4. Pure iridium and an alloy of 10% ruthenium and 90% iridium.

The first alloy is used for all temperatures up to 1600°C. The Constantan couples are useful for low temperatures between zero and 300°C. They have a high E. M. F. with the further advantage of being cheap. The silver and silver platinum couples are good for all temperatures up to the melting point of silver (926°C). This couple costs much less than the platinum and has nearly double the E. M. F. The ruthenium-iridium couple is useful only for high temperature measurements, those above the melting point of platinum. It is extremely brittle, but can be used up to 2100°C.

Principle of the Thermo-Electric Pyrometer.—Whenever two metals in contact are heated, an electric current is generated which is a function of the temperature. In a circuit composed of a number of metals, the resulting current is equal to the algebraic sum of the component currents. The E. M. F. of a couple is not altered when one or several metals are

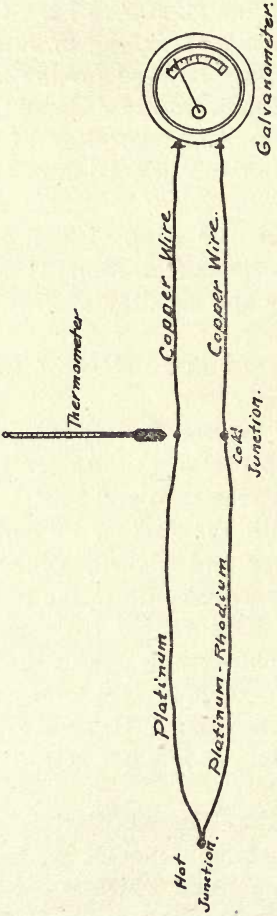

Fig. 18. THERMO-ELECTRIC PYROMETER

inserted between the metals of the couples. It follows that any metal can be used as a solder.

Before making the couples the wires should be annealed at a temperature equal to or even higher than that to which they will be exposed. The junction should be made by twisting the wires and melting the twist together in an oxygen blast until a globule forms. Care should be taken to avoid twisting and kinking the wires after annealing. Even in perfectly homogeneous metals, a distorted section will, when heated, give rise to an electric current which will of course affect the galvanometer to a greater or less extent. The thermo-couple should always be soldered to the copper leads, and never connected by merely twisting them together. The galvanometer may be placed at any convenient place, a mile removed if necessary. Care must be always taken, in order to insure accuracy, to standardize the couple with the same length of wires as that used for the work. The necessary length of copper may actually be used or an equivalent resistance.

It is to be remembered that the thermo-electric current generated is proportional, not to the actual temperature of the thermo-junction, but to the difference in temperature of the hot and cold junction. Care, therefore, should be taken to protect the connections of the thermo-junction with the copper wires (the cold junction) from undue heating; in case this protection is not possible it becomes imperative to use a water-cooled end. This is seldom necessary as it is usually sufficient to take the temperature of the cold junction as that of the surrounding atmosphere.

***Standardization of a Pyrometer.**—In order to find

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*Pyrometers and other instruments will be standardized for small fee at the Bureau of Standards, Washington, D. C.

a temperature corresponding to a given deflection of the galvanometer in millivolts, it is necessary to find the number of millivolts corresponding to known temperatures. Such temperatures are given by the melting or boiling points of chemical elements or definite chemical compounds. Mathematically speaking, with pure platinum and iridio and rhodio-platinum couples, it is sufficient to determine two points; by means of a logarithmic formula it becomes possible to find any other temperature corresponding to a given deflection. However, both for safety and rapidity it is better, in industrial work, to use the graphic method. If, on a sheet of co-ordinate paper, the temperatures are plotted on one axis and the millivolts on the other, the intersections of the ordinates and abscissae will give a series of points lying on a curve characteristic of the couple.

The fixed points to be used are as follows:

	Degrees C.
Boiling points of water, H_2O	100
" " " napthalene, $C_{10}H_8$	218
" " " sulphur, S	445
Freezing points of zinc, Zn	419
" " " aluminum, Al	655
" " " silver, Ag	962
" " " copper, Cu	1,084
" " " gold, Au	1,065
" " " platinum, Pt	1,780

For determining the deflections for the boiling points of water, napthalene, and sulphur, use the apparatus shown in Fig. 19.

The bulb test tube is inserted in a muff of plaster of paris, after having been filled to about one-third of its lower length with a substance to be used; heat is carefully applied until ebullition has begun. Care should be taken with sulphur and napthalene to keep the level of the vapors below the mouth of the test tube, otherwise the vapors will catch fire. For these determinations it is not usually necessary to protect the wires in the test tube; generally it is sufficient to spread them apart to prevent short circuiting the current. From the mouth of the test tube the wires may or may not be protected as is found necessary or convenient. The cold junction may be located in a bottle with a thermometer inserted in the cork.

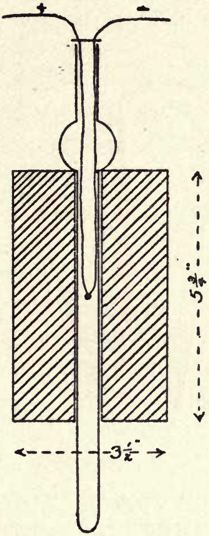

Fig. 19. APPARATUS FOR THE DETERMINATION OF BOILING POINTS

In ordinary work this is a useless refinement and the temperature of the cold junction may be taken as that of the surrounding room. Care should be taken to have the galvanometer sufficiently removed from the source of the heat. When reading the deflections for the boiling points, the needle of the galvanometer will reach a stationary position when the thermo-couple junction has reached an equilibrium of temperatures with the vapors. The pyrometer junction should be placed well within the plaster muff but not below the surface of the liquid. It is well to let the liquid cool and re-heat several times so as to furnish several observations. When couples other than the platinum groups are standardized, it is well to protect them from the vapors of sulphur

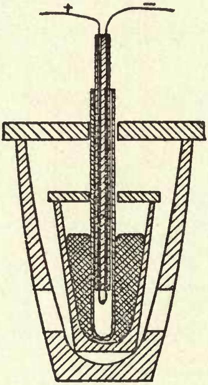

Fig. 20. APPARATUS FOR THE DETERMINATION OF MELTING AND FREEZING POINTS OF METALS

by means of a sealed hard glass tube as shown in Fig. 19. For the determination of the melting points or freezing points of metals the wires should always be protected; the best covering is afforded by porcelain tubes as shown in Fig. 20. About 35 to 40 grams of the substance to be used is taken and placed in a small crucible. No. 00 graphite crucible is well suited for the work. For zinc, aluminum, silver, and copper the crucible method is to be preferred, while for precious metals, gold and platinum, the wire method is the better. In the crucible method an apparatus is prepared as is shown in Fig. 20 by taking two crucibles fitting snugly one within the other. Two tuyere holes are cut in the outside crucible and the blast is supplied through the openings. The apparatus answers also well for the melting of gold and copper.

When making determinations it is well to insert the porcelain tubes at the beginning of the operation, before the flame is applied, so as to avoid cracking them by too sudden heating. The heating should be conducted slowly and continued until the metal is melted; this can be ascertained either by direct observation, by feeling the solid metal with the porcelain tubes, or by the deflection of the galvanometer. It is not necessary in the crucible method to watch the melting point. Once certain the metal is fused, the heating is stopped, the furnace is allowed to cool slowly and regularly and at the instant the mass has reached the freezing temperature, a well marked stop of the galvanometer needle is observed. With proper care and gradual cooling it is possible to have the needle stationary for 30 seconds with a mass of 35 grams of metal. It is always well to repeat the operation.

Temperature According to Color.—Approximate temperatures may be determined by observing the color. For this purpose the following table may be used:

Color.	Degrees C.
Lowest red visible in the dark	470
Dark blood-red or black-red	532
Dark red, blood-red, low red	566
Dark cherry-red	635
Cherry-red, full red	746
Light cherry, light red	843
Orange	900
Yellow	1,000
Light yellow	1,080
White	1,205

RETORTING

A good way to prepare amalgam for retorting is to wash and grind it in a Wedgewood mortar to remove the sand, then treat it with dilute nitric acid to remove dirt and copper, if present. A few pieces of metallic zinc agitated with the mercury and amalgam will often prove beneficial. Strain through ticking or chamois skin to separate the solid amalgam. The retorts are of various shapes and sizes. For laboratory use a half pint size as shown in the illustration is frequently used. Such a retort consists of an iron pot (a) with a cover (b) held firmly in place by a clamp (c) and a set screw (d). A bent iron tube (e) leads off the fumes, which are condensed with the aid of a water jacket of rags (f) and collected under water in a dish.

The retort should be thoroughly cleaned and coated on the inside with chalk, ruddle (Fe_2O_3), graphite, or better than any of those, first chalked on the inside and then lined with paper. This will prevent the residue from sticking to the bottom of the retort in case the heat is too high or zinc or lead is present. Place the amalgam in the retort, not over two-thirds full, smear the rim with a thick paste of flour, to make a tight joint, and clamp on the cover tightly. Heat very gently at first and gradually increase the temperature but never above a dull red, otherwise the retort will be bulged or melted. Mercury boils at 674°F. or 357°C. No fluxing material like borax should be used in the retort or the spher-

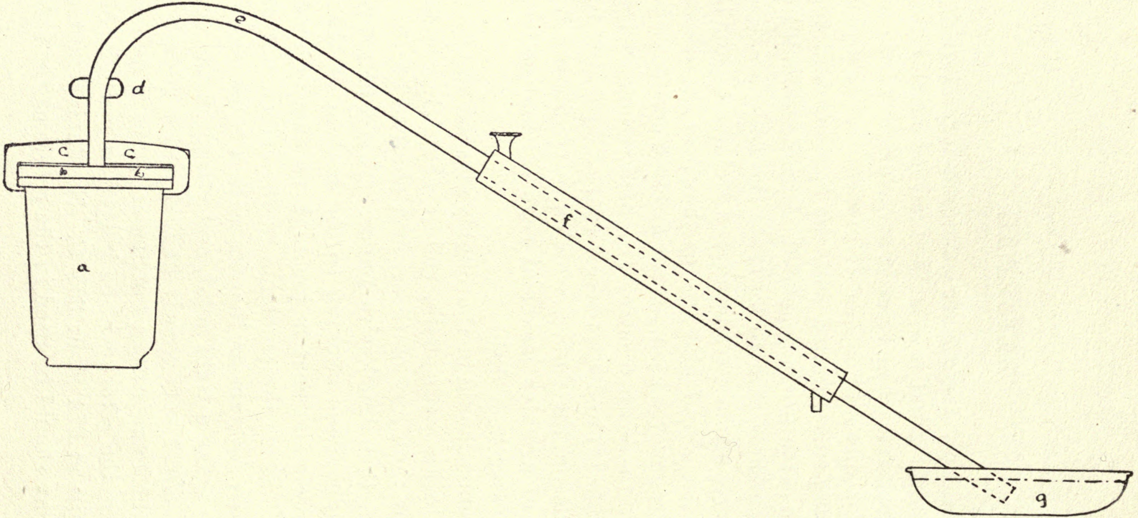

Fig. 21. RETORT.

oidal mercury will be changed to cohesive and boil with great violence. The residue might also be found permanently braized to the interior of the retort. Do not dip the delivery tube deep in the water. Just immerse the end of it in a small dish in order to prevent the possibility of water being sucked back into the retort by any sudden cooling and thereby causing a violent explosion because of the sudden generation of steam in the heated retort. Treat the resulting residue as given under 'Refining.'

For small amounts of mercury, where the gold alone is wanted, put the mercury-amalgam in a parting flask, after thoroughly cleaning, and dissolve off the mercury with nitric acid aided by heat. When completely parted, wash off the salts of mercury and acid with distilled water. If the residue contains foreign matter, filter carefully, so as not to lose any of the gold or residue, burn the filter paper in a lead tray weighing 10 grams, wrap the residue in the tray with enough silver to inquart the gold and cupel. The gold is now recovered by parting. If pure, the gold may be annealed and weighed at once without filtering or cupeling.

REFINING OF BULLION

If the bullion is base, nitre and borax glass are both needed in refining, but too much nitre will rapidly eat into the graphite crucible.

Lead, when present in the bullion, is best oxidized by nitre or sal-ammoniac; tin, by means of potassium carbonate; antimony and arsenic, by means of nitre or by stirring the bullion with an iron rod. Sodium carbonate is to be avoided unless there is silicious matter present, still a little of it with nitre seems to work well even if the bullion is quite pure. Bone-ash and silica save the crucible from the action of the oxides and are especially useful in thickening the slags in case thickening is necessary. If bullion is poured together with the slag it should be perfectly liquid.

Toughening serves to eliminate small quantities like arsenic, antimony, and lead, which render the bullion brittle and unfit for coinage or similar purposes. T. K. Rose recommends the addition of a little sal-ammoniac or corrosive sublimate to the melted bullion. Cover quickly to keep the volatile chloride fumes out of the room. Test by casting a small ingot, cooling and seeing if it will stand bending back upon itself without cracking.

Pure gold is a brilliant green color when melted and it may then be poured. Silver when nearly pure often bubbles violently in the crucible and especially so when much nitre has been used in the refining. The remedy is to lower the temperature, cover the melt with charcoal and stir with a graphite rod until bubbling ceases, then pour.

In pouring and casting, stir thoroughly and take a sample for assay immediately after. Cast in heated molds well coated with oil. Some use one-fourth inch of oil in the bottom of the mold, others sprinkle rosin in the bottom just before pouring. The object of the oil is to make a smoother ingot and by its burning on the top of the ingot to prevent all sprouting and tarnishing. Pouring small quantities of bullion is generally unsatisfactory and better results may be attained by letting the crucible and content cool. Break the crucible and clean the button, saving the slag for further treatment. Very small amounts of bullion may be refined by fusing with a charge as used in the regular assay and cupelling the resulting lead button.

In treating cyanide clean-ups, two methods are available.

1. Mix the wet slime with a little nitre, place on a tray in a muffle furnace and when dry increase the heat sufficiently to oxidize a large portion of the zinc. The oxidized charge may be smelted with the following flux:

	Parts.
Dry slime	100
Bicarbonate of soda	40
Borax	40
Silica	15

The bullion smelted by this method may be 800 fine.

2. In the acid treatment the zinc is dissolved in

a vat by adding sulphuric acid until all action has ceased when aided by heat. The slime is washed by agitating with boiling water and collecting in a filter-press. The following flux is an example of one used in smelting air-dried slime:

	Parts.
Slime	100
Borax	50
Bicarbonate of soda	30

The following is an example of a flux used when the slime contained lead from the use of the zinc-lead couple:

	Per cent.
Borax	60.0
Nitre	19.0
Sand	11.5
Soda	7.0

Manganese dioxide may be used to replace nitre, as it has a higher oxidizing power than the latter. Its use is not to be recommended with highly argentiferous slime as it tends to enrich the slag in silver.

Other methods used are scorification and cupellation in an English cupelling furnace; smelting in a small blast furnace to base bullion; retorting the zinc and fusion of the retort residues.

CONCENTRATION TESTS

It will be seen in the preceding chapters that concentration will often be a necessary part of an ore-treatment scheme. The following suggestions will prove of value in determining the methods to be followed.

PRELIMINARY OBSERVATIONS

Value and Character of Constituent Mineral.—Take a 50-gram sample of the ore and crush until the concentrate is free from the gangue. Separate the minerals from the gangue by hand-picking, panning, by using magnets, or with heavy solutions as the case may require. Each mineral taken out should be entirely free from gangue or other minerals. The total amount of each constituent is determined by analysis but the approximate amounts may be found from the weights separated. Assay each mineral for gold, silver, and constituent metal. It will prove useful to determine the specific gravities as well as to note the shapes and other characteristics of the grains. The value of the mineral to be saved will limit the closeness of the concentration and the elaborateness of the scheme to be followed. If the difference in specific gravity of the minerals to be separated is as much as two points, no especial difficulty will be found in the usual methods of water concentration, other conditions being favorable. If the products to be separated are of the same specific gravity, other methods must be followed. Zinc blende may be separated from garnet, rhodonite, and siderite by the Elmore vacuum oil process. Pyrite is removable from zinc blende by giving the mixture a flash roast and taking out the iron with a magnetic separator. In general, the closer the two minerals to be separated are in specific gravity, the closer must be the sizing for jigs and the classification for tables, and the more attention must be paid to re-crushing and re-treating the middling.

Friable minerals in general demand graded crushing, and graded concentrating or the removal of concentrate as soon as a sufficient quantity is liberated, as little screening as is consistent with good work, and the use of free-crushing machinery such as rolls. In many cases the valuable mineral is the most friable constituent of the ore, hence the necessity of free-crushing conditions. By graded crushing is meant the reduction of the lump ore to the final sizes by several stages of crushing and the removal of all undersize from the feed of each crusher. Screening can, however, be carried too far as sizing by this method produces slime by abrasion. By free-crushing conditions is meant the use of rolls, on account of the small amount of slime that they produce in comparison with other crushers. In other crushing devices, such as stamps, the conditions may be made very favorable by using a large screen discharge area with abundant water and low height of discharge. Concentrate should be removed as soon as liberated as further crushing will unnecessarily slime it, also because coarse concentration is cheaper and more effective than fine.

Determination of Included Grain.—Crush seven samples of the ore so as to pass through screens of the following sizes: 4, 8, 16, 20, 40, 80, and 100 mesh. Minerals may be liberated in sizes coarser than 4-mesh and, if this seems probable, larger sizes should be tried. Separate the concentrate by hand-picking or panning until entirely free from gangue in each case.

With the percentages of concentrate obtained in the above tests as ordinates, and the corresponding meshes as abscissae, plot a curve which will clearly show at what mesh the maximum amount of concentrate is liberated. The curve will show at what stage of the crushing to introduce concentration, how much finer the ore must be re-crushed before introducing another concentrating unit, to what degree of fineness all tailing or middling must be crushed to completely liberate the included grains, the probable recovery, and the percentage of concentrate in the ore. Additional value would be given to the above graph if the crushed samples were separated into clean concentrate, middling and clean tailing, and the data incorporated in the curve. A microscope will prove valuable in determining the percentages in the finer sizes.

SCREEN SIZING TESTS

In the examination of ores and mill products, it is often necessary to know the percentages of the different sizes present as well as their assay values. This may be learned by making a screen test. The screening is done by placing a weighed sample of the ore or product, say 100 to 500 grams, in the upper screen of a nest arranged in order of size, with the coarsest on top, and giving the whole bank a jarring, rocking motion. Before taking any screen from the nest it should be shaken separately for a definite length of time to make the last of the undersized grains pass through. In case of the finer sizes to start with, only a small portion of the ore should be placed on the screen at one time, because overcrowding destroys the efficiency of the screening. The ore resting on each sieve is weighed and if values are wanted, it is ground, if less than 100 mesh, mixed, sampled and assayed. The sizes of screens used might conveniently be 4, 8, 16, 20, 30, 40, 60, 80, 100, 150, and 200 mesh. These tests will be of value in finding where, or in what size the value lies in tailing or other mill products and thus suggesting a method of obtaining a higher extraction. They may be of use in determining the proportion of crushing to concentrating machinery, also in determining the efficiency of crushers, grinders, and screens. In considering the re-treatment of table middling resulting from a classified feed, a screen test will show if it would be of advantage to screen out and reject the larger and necessarily low-grade particles before further concentration or treatment.

WET CONCENTRATION

Preliminary Tests.—Concentrate coarser than ¾-in. in diameter may be efficiently removed by hand-picking. Waste rock may be removed in the same operation. For sizes down to 12 or 16 mesh a jig will be the most efficient means of recovering the concentrate but on the finer sizes a table does better work.

Jigging Tests.—To feed a jig, so as to obtain the most efficient work the material should be screen-sized or at least hydraulically classified. On account of the metallic minerals being present in the classified feed in smaller particles than those of the gangue, the product is better suited to hutch work when jigging. The proper range of sizes to be treated on one jig is limited by the hindered settling ratios of the minerals to be separated.

HINDERED SETTLING RATIOS OF QUARTZ AND COMMON MINERALS*

Quartz and—	Ratio diam. quartz, to diam. mineral.
Copper	8.6
Galena	5.8
Wolframite	5.2
Antimony	4.9
Cassiterite	4.7
Arsenopyrite	3.7
Chalcocite	3.1
Pyrrohotite	2.8
Sphalerite	2.1
Epidote	2.0
Anthracite	0.2

*Taken from Richards' 'Ore Dressing,' Vol. I.

Take as an example the minerals quartz and galena of which the hindered settling ratio is 5.8. This means that a particle of quartz 5.8 times as large as a particle of galena will be found in the same position or layer in a jig bed. Hence, to obtain a perfect separation of clean concentrate from clean tailing, the range of sizes to be treated on one jig must be somewhat less than the hindered settling ratio. Middling or included grains will assume positions according to the percentage of included concentrate and must be re-crushed and re-sized for further treatment. In the coarser sizes, all the tailing and middling may require re-crushing, as is the case with some of the Butte ores.

In running a jig for concentrate only, a screen bed is used in which the opening is slightly smaller than the smallest grain in the feed. With the finer sizes, 10 to 20 mesh, a 16-mesh screen would be used with a ½-in. bed of coarse concentrate (4 to 10 mesh), for the purpose of making hutch product and concentrate. In separating the finer sized particles, screening will often prove impractical, in which case, a hydraulic classifier, acting under free settling conditions, should replace the last screen or trommel.

A 25 to 50-lb. sample of the ore is crushed as nearly as possible under conditions that would be used in practice. The coarser sizes are screened out for the jig work and the undersize saved for further treatment. Each lot or size, which has been previously weighed and assayed, is run over a jig of the Vezin laboratory size and the material separated into concentrate, middling, hutch product (if any), and tailing. Weigh and assay each. The assay values of the middling and tailing will determine whether they are to be rejected or treated further.

Fine Concentration.—This work may be done roughly in a miner's gold pan or under conditions more nearly resembling those in a mill on a laboratory size table of the Wilfley type. The products treated may include the undersize from the trommels whose oversize is fed to jigs, re-ground middling or tailing from the fine jigs, re-ground table middling, or all the product, if it has been crushed at once finer than 16 mesh. To secure the best results, the feed to a table should be hydraulically classified into two or more sizes according to the coarseness of the original feed or the difficulty of the separation. As many as six sizes are used. The pulp should not be too liquid, hence it should be de-watered before sending to the classifiers or feeding to the tables. The overflow from the classifiers and the de-watering boxes should go to a final large settling tank whose spigot product is treated on slime tables. Each lot or size, which has been previously weighed and assayed, is run over a laboratory table of the Wilfley type and separated into concentrate, middling, and tailing. Weigh and assay each. The further treatment of the middling is dependent upon its contained values.

The tailing from the tables treating the finer sizes as well as the overflow from the settling tank, may be further treated by running over a small canvas table. One 10 ft. long and 2 to 4 ft. wide, covered with No. 6 canvas duck, will be found convenient. The woof or cross threads are laid down the slope as the concentrate is more effectually caught in the higher ridges by the warp.

Draw up a report which will include the percentage of recovery of each separate run or test based on, (a) the total value of the feed to each test, (b) the total value of the total weight of ore taken for the experiment. The total values accounted for in the total concentrate and the tailing subtracted from 100% will give the losses due to slime and handling. Give the probable final recovery and a flow sheet of the treatment followed.

MAGNETIC SEPARATION

This process is based on the fact that certain minerals possess magnetic qualities which permit their being lifted out of the attendant gangue by the action of electromagnets. By roasting at a dull red heat for a short time, certain non-magnetic iron minerals may be rendered magnetic and thus a separation may be effected. This fact is used in the separation of pyrite from zinc blende, which are taken off together from the concentrating tables and jigs.

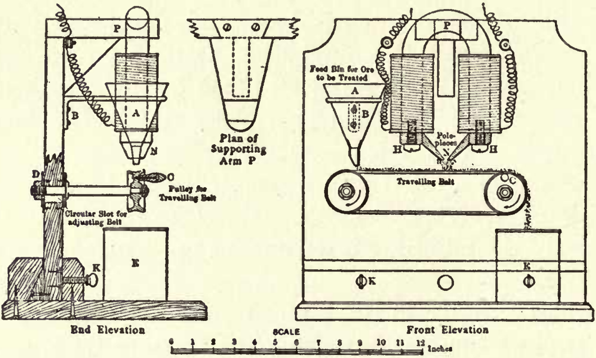

Fig. 22. ELECTROMAGNET FOR TESTING ORE

(After L. H. L. Huddart, *The Engineering and Mining Journal*)

For effective work the ore fed to the separator must be sized. The closeness of sizing depends upon the similarity of the magnetic properties of the minerals to be separated.

Test.—Assay the original ore for gold, silver and metal constituents. Take a definite weight of sample, say 500 grams, place in a cast-iron roasting-dish and heat in a muffle or oven-furnace at a dull red heat until the flames of burning sulphur have entirely disappeared. Cool down by withdrawing from the furnace, and weigh. Note the change in weight due to the roast.* Remove the magnetic mineral by carefully passing over the machine shown in Fig. 22. The material should be treated two or three times. The separation may be effected by spreading out the roasted ore on a clean level sheet of paper in a thin layer and removing the magnetic mineral with a strong horseshoe magnet. Weigh and assay both the concentrate and the tailing. Calculate the percentage of recovery or the effectiveness of separation as the case may require.

ELECTROSTATIC SEPARATION

This process is based on the *difference* in electrical conductivity and inductivity of minerals in an electrical field, and also the fact that bodies charged with like signs, or kind of electricity, repel, and with unlike signs attract each other. Every mineral if subjected to a sufficiently high electrical pressure becomes a conductor to an extent depending upon the

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*This roast may be omitted if the constituent to be separated is naturally magnetic.

mineral. The fundamental principles may be utilized in various ways, of which two are given: (A) The mineral mixture, in a neutral electrical condition, is brought into contact with a highly charged surface, whereupon the best conductors are expelled first and poorest last. (B) The mineral mixture with a charge of one kind of electricity is brought into contact with a surface having a charge of the opposite sign, whereupon the good conductors will leave the surface first, and the poorest is attracted and held the longest, by reason of its reluctance to change its charge of an unlike sign to a like sign.

The following is a list of minerals placed according to their conductivities. This list indicates clearly the possibilities of the process, the basic principle stated first being recalled.

GOOD CONDUCTORS

- *Most sulphides*
  - Pyrite
  - Chalcopyrite
  - Galena
- *Native metals*
  - Copper
  - Gold
- *Some oxides*
  - Magnetite
  - Hematite

POOR CONDUCTORS

- *Quartz*
  - Quartzite
  - Sandstone
- *Feldspars*
  - Granite
  - Porphyry
  - Andesite
- Epidote
- Garnet
- Calamine
- *Most carbonates*
  - Calcite
  - Siderite
  - Limestone
- *Most sulphates*
  - Barite
  - Gypsum
  - Sphalerite (zinc blende)

The principal application of this process is, at present, the separation of zinc blende from other sulphides where water concentration fails on account of the slight differences in specific gravity. There may be thus produced from a zinciferous concentrate of low or doubtful value because of the presence of zinc, a high-grade zinc product and a very much improved concentrate of the remaining sulphides. Electrostatic separation might also be used to effect a dry concentration of sulphide ores from quartz and other gangues, where water is scarce; for the concentration of easily slimed ores, such as the silver sulphides; for the separation of high specific gravity minerals, such as garnet, barite, and epidote, from the valuable sulphides; for the concentration of molybdenite, graphite, and monazite, as well as other minerals where the differences in electrical conductivities open a way.

The process should be taken into account where the simpler methods of water concentration are not successful because of commercial reasons, such as water supply or because of the physical nature of the mineral. It may also be used in conjunction with water concentration.

The experiments should be conducted with a commercial sized electrostatic unit, under such rules as would govern a regular concentration test. The Huff Electrostatic Separator Co. make a laboratory unit for testing 3-pound lots of ore.

OIL FLOTATION

The Elmore vacuum process is based primarily upon the fact that oil has the power of wetting certain metallic mineral particles as distinct from the gangue. The selective action is further increased in some cases by the fact that gas bubbles liberated by an acid and aided by a partial vacuum, attach themselves to the greased particles thus causing them to float. The process is applicable to the separation of chalcopyrite from magnetite and spathic iron gangue, galena and zinc blende from baryta, garnet and similar heavy gangues. Friable minerals such as sulphides of antimony and molybdenum that suffer high losses in water concentration are found amenable to concentration by this process. The process does not lend itself readily to simple and inexpensive laboratory experiments. Samples should be shipped to the manufacturers of the apparatus for testing.

DETERMINATION OF THE SUITABILITY OF AN ORE FOR SMELTING

Preliminary estimates of the profit or loss in smelting an ore are conveniently made by applying a schedule of smelter charges that would be used for the territory in which the ore occurs. This method does not require the determination of the most economical blast-furnace charge for smelting the ore under the conditions existing in the region in question or the cost of fluxes, fuel, labor and construction of a plant, since all these items are embodied in the smelter schedule as the results of experience with the ores in the vicinity. In cases where there is no competition, the charges and penalties might be excessively high and the payments for metals correspondingly low, which must be carefully considered when making estimates. In figuring the final profit, allowances must be made for mining and handling.

The schedule may be used for determining the advisability of smelting the ore raw or concentrating first, as well as the limit or cleanness of concentration, by applying the schedule to a given amount of raw ore, say 100 tons, and to the concentrate obtained from an equal amount of ore. In the same way the limit of concentration may be determined. An analysis of a fair sample must be made for gold, silver and other substances that are penalized or rewarded under the proper schedule.

SCHEDULE

FOR DRY ORES, CONCENTRATE AND 'TAILING,' LEAD ORES, AND LEAD CONCENTRATE, CLEAR CREEK AND GILPIN COUNTIES, COLORADO

February 1, 1905

All rates f.o.b. cars, Denver.

DRY 'TAILING' AND CONCENTRATE

GOLD, $19 per oz., if $^{5}/_{100}$ oz. or over per ton.

SILVER, 95% of N. Y. quotation day of assay, if 1 oz. or over per ton.

COPPER, dry assay (wet less 1.5 units).*	Per unit.
5% or less	$1.25
Over 5 and including 10%	1.50
Over 10%	1.75

10% silica basis, 10c. charge for each per cent over 10.†

5% zinc basis, 30c. charge for each per cent over 5.

When gross value is:	Treatment charge.
Not over $35 per ton	$3.50
Over $35 and including $80 per ton	4.00
Over $80 per ton	5.00
Upon lots of less than 7 tons	5.00

*A unit is 1 per cent.

†Subtract silica from iron, *i. e.*, an ore containing 30% SiO_2 and 15% Fe would be figured at $30 - 15 = 15\%$ SiO_2, and *vice versa.*

DRY SILICIOUS AND COPPER ORES

GOLD, $19.50 per oz., if $^{5}/_{100}$ oz. or over per ton.

SILVER, 95% of N. Y. quotation, day of assay.

COPPER as in schedule of concentrate.

$8 treatment charges.

40% silica basis, 5c. credit for each per cent less than 40, and 10c. charge for each per cent over 40, up to a maximum charge of $11 on ores not exceeding $25 gross value; and $12.50 on ore exceeding $25 gross value.

5% zinc limit, 30c. charge for each per cent over 5.

OXIDIZED IRONY ORES

GOLD, $19 per oz., if $^5/_{100}$ oz. or over per ton.

SILVER, 95% of N. Y. quotation, date of assay.

LEAD, 25c. credit per unit for 5% or over.

$2 treatment charges.

Neutral basis, 10c. per unit charge for each per cent silica excess.

LEAD ORES

GOLD, $19.50 per oz., if $^5/_{100}$ oz. or over per ton.

SILVER, 95% of N. Y. quotation date of assay.

LEAD, prices flat.

COPPER, $1 per unit dry (1.5% off wet) when ore assays 2% wet.

ZINC limit 10%, 50c. charge for each per cent over 10.

NEUTRAL SCHEDULE

Assay, per cent Pb inclusive.	Cents per unit.	W.C.*
5 to 10	25	$8.00
Over 10 " 15	25	7.00
" 15 " 20	25	5.00
" 20 " 25	25	4.00
" 25 " 30	30	4.00
" 30 " 35	30	3.00
" 35 " 40	30	2.50
" 40 " 45	32	2.00
" 45 " 50	35	2.00
" 50	40	2.00

Neutral basis, 10c. credit for each per cent iron excess, and 10c. charge for each per cent silica excess.

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*Working charge.

FLAT SCHEDULE

Assay, per cent Pb inclusive.	Cents per unit.	W. C.
5 to 10	25	$12.00
Over 10 " 15	25	10.50
" 15 " 20	25	8.50
" 20 " 25	25	6.50
" 25 " 30	30	6.00
" 30 " 35	30	4.50
" 35 " 40	30	3.00
" 40 " 45	32	2.00
" 45 " 50	35	2.00
" 50	40	2.00

Neutral schedule to be used when it figures better for the shipper.

LEAD CONCENTRATE

GOLD, $19 per oz., if $^5/_{100}$ oz. or over per ton.

SILVER and COPPER as in lead ores.

LEAD, prices flat.

SILICA limit, 10%, 10c. charge for each per cent silica excess over 10.

ZINC limit, 5%, 30c. charge for each per cent zinc excess over 5.

Assay, per cent Pb inclusive.	Cents per unit.	W. C.
5 to 10	25	$4.75
Over 10 " 15	24	4.00
" 15 " 20	30	3.50
" 20 " 25	32	3.25
" 25 " 30	35	3.25

Upon concentrates assaying over 30% lead, apply 'Neutral schedule' or 'Flat schedule,' whichever figures better for the shipper; $19 for gold.

SCHEDULE

FOR ORES AND CONCENTRATE, BOULDER COUNTY AND CRIPPLE CREEK, COLORADO

February 1905

All rates f.o.b. cars, Denver.

GOLD, $19 per oz., $^5/_{100}$ to 2 oz. inclusive per ton.

GOLD, $19.50 per oz., if over 2 oz. per ton.

SILVER, 90% of N. Y. quotation if ore assays from 1 to 10 oz. per ton.

SILVER, 95% of N. Y. quotation if ore assays over 10 oz. per ton.

Gross value.	Treatment.
Up to $10	$4.00
Over 10 to 20	5.00
" 20 " 30	5.50
" 30 " 40	6.00
" 40 " 50	6.50
" 50 " 75	7.00
" 75 " 100	8.00
" 100	9.00

When ore does not exceed $10 gross value, 3%, sulphur limit, 25c. charge for each per cent of sulphur over 3, up to a maximum charge of $2.50 per ton, zinc limit 5%, then 30c. charge for each per cent of zinc over 5.

When ore is over $10 gross value, no sulphur limit, zinc limit 5%, 30c. charge for each per cent of zinc over limit.

LEAD ORES

Apply schedule for lead ores for Clear Creek and Gilpin counties.

CONCENTRATE, LEAD OR DRY

Apply corresponding schedule for Clear Creek and Gilpin counties.

PHYSICAL PROPERTIES OF SLAGS

General.—The melting point of a slag and the superheat necessary above the melting point to make it flow properly may be determined by experiment. A complete analysis of the slag should be made and the silicate degree determined. The physical characteristics visible to the eye, that will aid in the approximation of the analysis, are also to be investigated.

Take a 500-gram sample of the granulated slag and make a complete analysis. Note the luster and appearance of surface and fractures of the granules. Determine the specific gravity.

Melting Point Determination.—Take a 50-gram sample and grind so as to pass a 100-mesh sieve, being careful not to contaminate with foreign substances. Moisten thoroughly with a heavy lubricating oil in which 10 to 15% vaseline has been dissolved by the aid of heat. Ram the material into a slag cone mold as shown in Fig. 23, in thin layers, scratching each surface before adding the next lot to provide a good bond. Use a rammer ¼ in. square and ram each layer very hard before adding the next. Remove the cone from the mould by unclamping and sliding the halves past one another. Dry by setting in front of a muffle and raise slowly to a dull red heat, burning off the binding oil. The cone is now ready for testing. Place the dried cone on a chromite or graphite slab which may be conveniently bedded in bone ash in a scorifying dish for handling, and place in the interior muffle of the gas forge. The doorways with peep-holes are placed in position over the mouth of the muffle of the gas forge. The pyrometer is next inserted through the top holes so as to have the hot junction as near the slag cone as possible.

Light the gas and regulate the blast so as to take one-half hour in bringing the muffle to a dull red heat. In the next hour raise slowly past 1000°C. to the melting point of the cones, observing the melting temperature of the cones by looking through the peep-holes. Only remove the inner doors for observation if it is impossible to see the cones on account of the uniform color and temperature of the slag cones and interior of the muffle.

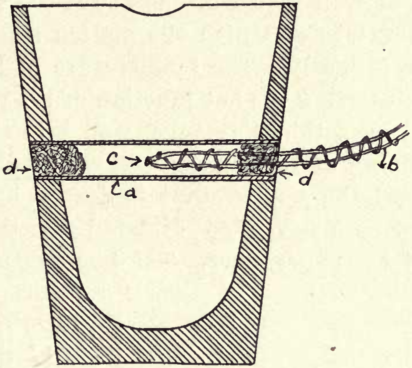

Fig. 23. CRUCIBLE FOR SLAG TESTS
a Thin fire-clay tube; *b* thermocouple; *c* hot junction; *d* asbestos plugs.

Necessary Superheat and Appearance When Molten.—Fill with granulated slag a fire-clay crucible, lined with a coating of chromite and water-glass; if necessary on account of a basic slag, melt down in a gas forge-furnace, until thoroughly liquid. Dip a cold iron rod into the slag and note the appearance of the drops falling from the rod, that is, whether they string out or break off short. Note the appearance of a stream of slag as it is being poured into a bucket of water. By means of several trials note the temperature at which the molten slag is liquid and flows properly. The temperature of the slag is taken by inserting the hot junction of the pyrometer, protected by a thin clay tube, well into the slag at the beginning of the experiment and having an observer read the temperature indicated by the galvanometer, whenever so directed by the one in charge of the experiment. The temperatures are to be recorded, with appropriate remarks. Pour some slag into a mould and note appearance of surface and fracture.

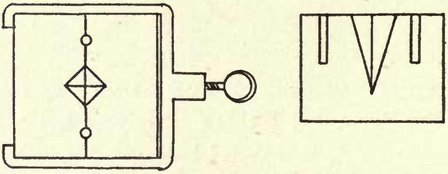

Fig. 23. SLAG CONE MOLD

Report.—(1) analysis of slag; (2) specific gravity; (3) silicate degree; (4) melting temperature, T_1;

(5) temperature of slag when liquid, T_2; (6) superheat; (7) appearance of surface of molten slag when pouring, also nature of slag, comparative viscosity, tests with cold iron rod, and general remarks; (8) give commercial reasons and conditions that would require the use of this slag.

This test may be varied to show the characteristics for any particular type of slag, irony, zincy, limy, aluminous, acid, or basic.

SMELTING PROPERTIES OF COPPER, LEAD, GOLD, AND SILVER ORES

The object of this chapter is to answer the question: Given an ore or collection of ores, what are the metallurgical conditions to be attained to result in their most profitable treatment? The ultimate purpose of the investigation may be for a comparison of fire and wet methods, or for determining the commercial value of an ore. It is evident that the sum of the assay values will not answer this last question. There are many factors which enter into the final smelting cost of an ore, the chief one being its metallurgical behavior. In general it is desired to smelt an ore with the least possible addition of fluxes, as is compatible with the greatest net profit. Fluxes not only cost money but also take up valuable space in the furnace, reducing its capacity for the profit-yielding portion of the charge and thus increasing smelting costs. When fluxes contain metals of value and are subject to treatment charges, this fact modifies the above statement, but then the flux would rightly be classed as an ore. On the other hand there are disadvantages resulting from insufficient fluxing. The idea is to use the minimum amount of flux that is allowable in view of the increased cost of coke and lower furnace capacity with highly silicious and limy slags. The increased loss of metal from imperfect separation due to an excess of the basic oxides in slags of high specific gravity must also be reckoned with. In the case of ores containing undesirable substances such as Al_2O_3, MgO, BaO, $BaSO_4$, and ZnO or ZnS, it must often be considered to what extent the charge will have to be diluted with low-grade or unprofitable ores and fluxes in order to bring the mixture within smelting range. The function of the following tables and tabulated information is to convey to the student the limits to which he may go with a slag when seeking to estimate the greatest probable net profit, and also to show the effect of an increase or decrease of a charge constituent upon a metallurgical result.

LEAD SLAGS

In lead smelting the operator is confined in the beginning to the selection of charges by the comparatively rigid limits of the regular type slags, a number of which are given in the following tables. By this statement is not meant that the so-called 'type slags' are the only ones that will work properly but rather that certain slags which have been tried are known to be satisfactory. The metallurgist may often have to sacrifice what he considers good metallurgical practice to obtain the greatest net profit, hence the type slags are not always followed. In the following table the analyses of type slags are given with the remarks on each. Under the columns headed MnO, MgO, BaO, etc., are given the maximum percentages of these substances that have been used with the corresponding type. By this it is not intended to mean that the entire list of impurities would be allowable in one slag to the extent of the maximum figures given. For the maximum percentages of one impurity in the presence of others, the reader is referred to the explanations following the table. A slag is designated as whole, half, quarter, etc., according to the ratio of the FeO + MnO to the CaO + MgO + BaO. Thus in a half-slag the CaO, etc., is approximately one-half the percentage of FeO. When the MgO, etc., is added to the CaO it is added according to its replacement value; thus 40.3 lb. of MgO is equivalent in fluxing power to 56.1 lb. of CaO. By the word 'charge' as in 'per cent lead on the charge,' is intended the blast-furnace charge including ore and fluxes but not the coke or slag for re-smelting. In the 'Table of Limits' is given the maximum, minimum and average figures for slag constituents found in practice, also figures for other data important to silver lead smelting. Unusual figures are enclosed in brackets.

SLAG TYPES

		Per Cent.			Limits with Corresponding Type Slag. Per Cent.				
		SiO_2	$FeO + MnO$	$CaO + MgO + BaO$	MnO	MgO	BaO	ZnO	Al_2O_3 ......
A	1	35	28	28	5	5	2-5	10	6-10
B	1	36	31	23	5	5	2-5	10	6-10
	2	34	34	24	5	5	2-5	10	6
C	1	35	38	17	5	5	2-5	8-15	6
	2	31	38	21	5	5	2-5	8	6
D	1	30	40	20	8	5	5	8	5
E	1	32	47	11	8	5	5	8	5
	2	30	48	12	8	5	5	8	5
F	1	28	50	12	8	5	5	8	5
	2	28-30	54	6	8	5	5	8	5

REMARKS

A—Useful in case of an excess of silicious ore and for ores containing much alumina. It is not adapted to ores high in zinc.

B—More fusible and faster running than A, but still too silicious for ores high in zinc.

C—This type may be used successfully to work off zincy ores.

D—An excellent slag adapted to working irony ores; also good for zincy slags.

E—A good slag for working off irony ores, where they are in excess or most profitable to treat.

F—Not as good as the preceding types and of rather high specific gravity. Only used in case of an excess of irony ores and where silicious fluxes are not available at a reasonable cost.

SLAG CONSTITUENT

	Per cent.		
	Maximum.	Minimum.	Average.
[1]Silica, SiO_2	36	28	30-34
Iron and manganese, $FeO + MnO$	54	28	30-45
Lime and magnesia, $CaO + MgO$	28	6	12-24
Manganese, MnO	43	0	8-25
Magnesium oxide, MgO	5	0	3- 4
Barium oxide, BaO	8	0	0- 5
Zinc oxide, ZnO	15	0	0- 5
[2]Alumina, Al_2O_3	10	0	0- 7
Sulphur, S	1.5	0.8	1.0
[3]Lead, Pb	1.5	0.5	0.8-1.0
[3]Silver, Ag	2 oz.	0.5 oz.	
[3]Gold, Au	0.01 oz.	Trace	

GENERAL DATA

[4]Zinc, volatile	20	10	.....
Sulphur, volatile	30	20	20
[5]Fuel (coke)	17(22)	13	14
Lead in matte	18	7	12-14
Copper plus iron in matte	60	55	.....
Copper in matte	0	16	8-12
Sulphur in matte	23	20	.....
[6]Matte fall	20	(4-7)6-8	8-10
Scrap iron on charge	10	5	.....
Assay base bullion—			
Silver, Ag	300 oz.	90 oz.	.....
Gold, Au		0.1 oz.	
Lead on charge	20	6	10
Recovery of lead	95	90	.....
Recovery of gold	98	..	.....
Recovery of silver	98	95	.....

[1] Successful runs have been made with 43% SiO_2.

[2] 8.5% is too high for speed in smelting with a slag containing SiO_2 32, Fe + Mn 25%.

[3] An actual slag contained: Pb, 0.8%; Ag, 1.0 oz.; Au, 0.004 oz.

[4] Depending upon completeness of roast.

[5] Figured on the basis of good firm coke with 10% ash.

[6] Ratio of matte to ore plus flux.

EFFECT OF SLAG CONSTITUENTS

Alumina.—Aluminous slags will generally be slow running. They require a higher heat to make them flow properly than their temperatures of formation would indicate. Iles recommends that the iron content be kept well advanced and the silica be between 29 and 31 per cent.

Baryta.—This enters both the slag and matte and increases the specific gravity of slag so as to make proper settlement of matte difficult, otherwise a good alkaline earth flux. Baryta ores have been successfully handled by using a slag running SiO_2, 35 to 37%; Fe, etc., 20; lime, 27; BaO, 14.

Lime.—In general high lime makes slags low in lead except where zinc conflicts and requires the iron to be increased.

Magnesia.—Magnesia is especially troublesome in the presence of zinc and alumina, forming streaky, pasty slags high in lead. In a slag containing 8% zinc and 2 to 3% baryta, MgO is troublesome and 5% will give serious trouble. Magnesia alone and under 5 to 8% will give little trouble when intelligently handled. It has 1.4 times the replacement power of lime.

Potash and Soda.—These elements in excess will cause slags to run high in lead but good results may be obtained by using a hot slag containing 17 to 22% of alkaline earths.

Zinc.—Zinc decreases the fluidity of the slag and, while it lowers the temperature of formation, more fuel is required to make the slag flow properly and keep the lead content down. Zinc enters the matte and decreases the specific gravity making it difficult to settle properly. An excess of zinc is best taken care of by irony slags as indicated in the table. It is customary to preserve the slag type by replacing lime with one-half the zinc oxide present, and adjusting all the constituents so as to add up to the percentage of the original total. The amount of zinc volatilized depends upon how well the ore has been roasted and the zinc changed into oxides; a poorly roasted ore will result in a low volatilization of zinc. The amount of zinc volatilized and going into the matte will run from 10 to 20%. Following is example of slags, etc., resulting from smelting zinc retort residues:

Base bullion.		Slag.		Matte.	
Ag	87.5 oz.		Per cent.	Ag	19.0 oz.
Au	1.49 oz.	SiO_2	31.1	Au	0.04 oz.
		FeO	37.5	Pb	8.7%
		MnO	1.5	Cu	1.5%
		CaO	14.1		
		ZnO	10.0		
		Pb	0.77		
		Ag, oz.	1.26		

FURNACE DATA

	Per cent.		Saving, per cent.
Lead on charge	9.1		
Coke on charge	13.0	Silver	90.8
Slag on charge	32.6	Gold	92.4
Sulphur on charge	3.7	Lead	92.0
Matte formed	7.2		

GENERAL REMARKS

The slags to be re-smelted, consisting regularly of the shells from the slag dump-pots and any foul slag from clean-ups or unfavorable runs, will average about 20 to 30% of the total 'charge,' and even 35 to 40%; the term 'charge' referring to ore fluxes, but not the coke or slag for re-smelting.

Iron.—Where iron fluxes are dear a good slag would be: SiO_2, 32%; Fe, 32; CaO, 22; Zn + Al_2O_3, 10.

Base Bullion.—There should be enough precious metal on the charge so that the silver content of the base bullion will run between 100 and 300 oz. per ton and at least 0.10 oz. of gold per ton, as the refiners pay only for 0.05 oz. Au or over. The richer the bullion the greater the amount of precious metal that will be found in the slag and matte. A typical base bullion contained, Ag 266 oz., Au 3.49 oz., Pb 95.0%, As 0.28, Cu 0.70.

Lead.—For efficient collecting of the values, the percentage of lead on the charge should not be under 6 and better 10. Over 20% is liable to force too much lead into the slag, though in one smelting works as high as 30 to 35% lead has been placed on the charge and the resulting slags have contained less than 1% lead under favorable conditions.

Fuel.—The quantity of fuel is lowered by high sulphur and a high percentage of lead on the charge, also by an easy running slag having a low temperature of formation. The maximum fuel under these conditions has been as low as 9 to 10% of coke.

Matte Fall.—The quantity of matte formed will depend upon the amount of sulphur available after volatilization and slagging. The slag will contain about 1% sulphur and the volatilization will be from 20 to 30% depending upon the condition of the ore; an unroasted ore will give a higher loss of sulphur than a roasted product. The matte fall should average about 10% in order to keep down the silver-lead content of the slag, and allow the handling of more impure and refractory ores than the lower matte falls given in the table. Low matte fall may be used with pure silicious ores in the absence of $BaSO_4$. It will also depend upon the cost of roasting and re-smelting the matte. Where manganese is substituted for iron in the charge it lessens the production of matte.

Matte Content.—The matte should ordinarily contain not less than 5 and not more than 14% copper to effectually cleanse the slag of silver. When in excess of 14 some of the copper is reduced and goes into the base bullion where it must be removed by drossing and returned to the blast-furnace. Since the matte is roasted and returned to the furnace, the copper accumulates and when in excess of the limit desired, it is removed and smelted separately to form a converting grade of copper matte. The lead, which is mostly volatilized in this process, is collected in the bag house. The following charge may be used for the concentration: (1) matte roasted to 4% sulphur; (2) silicious ores as free from S, Au, and Ag as possible; (3) copper ores; (4) limestone; (5) slag; (6) coke, 13 to 14%. The slag should run between 35 and 42% SiO_2. The matte formed should contain about 50% Cu and 15 to 20% Pb. Scrap iron may be added to the charge to the extent of 5 to 10% to lower the lead content of the matte, which should ordinarily run between 12 and 14% but can be reduced to 7 or 8% should the conditions warrant. It is not advisable to use scrap iron when there is much zincblende present, because metallic zinc is reduced and oxidizes to ZnO in the upper part of the furnace, forming accretions and preventing free combustion of the coke from the coating of oxide upon it. A lack of fuel to sufficiently superheat the slag and give proper reducing conditions is quite apt to produce a matte high in lead. In general a slag of 25 to 28% FeO is best suited for producing a matte low in lead. The lead also increases with the copper in the matte, namely, a 40% copper content may call for 20% lead. A typical matte analysis is: S, 20.29%; Fe, 35.0; BaO, 0.73; Zn, 6.42; Pb, 10.96; Cu, 14.8; Ag, 77 oz. per ton; Au, 0.18 oz. per ton; specific gravity, 4.64.

Speiss.—When sufficient arsenic is present to form a speiss, 2.3 times the weight of arsenic equals the weight of iron needed to form it. This must be subtracted from the total iron in the charge as it is not available for slag or matte. The per cent of speiss formed may be approximated by running a regular lead assay on the ore and noting the weight of the speiss button found with the lead. Ores forming much speiss should be run with a silicious slag in order that the temperature of the furnace may be high enough to keep the speiss liquid.

Losses.—In best practice 90 to 95% recovery of the lead and 98% of the gold and silver can be obtained.

Flue Dust.—The amount of flue dust will run from 0.3 to 3.5% (and more) of the total charge according to the amount of fine material and the strength of the blast. An average figure would be 2½ per cent.

Lead Roasting.—The question when to roast will be decided by the matte fall desired. Sulphur is commonly debited at 25c. per unit, which represents the approximate cost of handling and reworking the resulting matte. Then, since for each unit of sulphur driven off in a roast, 25c. is saved, the advisability of roasting may be estimated by taking into account the cost of roasting and the resulting losses which are from 2 to 7% of the lead and 1 to 5% silver, but can be kept within 3 to 4% lead and 1 to 2% silver with a trace of gold. The change in weight of a roast approximates the replacement of sulphur by oxygen and also in the case of a pot roast the loss of CO_2 of the limestone when used.

COPPER SLAGS

The copper slags are usually classed according to the ratio of the oxygen on the base side to the oxygen on the acid side. Iron and lime oxides being the chief bases and silica the principal acid.

(1) Subsilicate .... $3ROSiO_2$
Ratio 3 to 2.
(2) Singulosilicate .... $2ROSiO_2$
Ratio 2 to 2.

(3) Sesquisilicate .... $4RO3SiO_2$
Ratio 2 to 3.
(4) Bisilicate .... $ROSiO_2$
Ratio 1 to 2.
(5) Trisilicate .... $2RO3SiO_2$
Ratio 1 to 3.

In these equations RO represents the oxide of any base or combination of bases.

The silicate degree of a complex slag is found by taking the percentage of oxygen in the SiO_2 or acid side and that on the base side by the use of the following table and determining the ratio.

	Per cent oxygen.
Silica, SiO_2	53.35
Iron oxide, FeO	22.2
Manganese oxide, MnO	22.6
Lime, CaO	28.6
Magnesia, MgO	40.0
Baryta, BaO	10.5
Alumina, Al_2O_3	47.0

For example, take slag of the following analysis:

	Per cent.	Per cent O on base side.	Per cent O on acid side.
SiO_2	40.0	...	21.3
FeO	30.0	6.7	...
MnO	5.0	1.1	...
CaO	20.0	5.7	...
MgO	5.0	2.0	...
		$15.5 = x$	$21.3 = y$

$x:y$ as $15.5:21.3 = 1:1.37$; approximately 3:4 or $3RO,2SiO_2$.

The **subsilicate** would only be used in case of a great excess of basic ores, usually irony, with silicious fluxes not available at a reasonable cost. This type requires a rather high temperature to run properly; they are very corrosive to crucible linings, and are of too high specific gravity to make a clean slag, and hence are generally unsuited to economical smelting except in extreme cases. The **singulo silicate** enters the range of ordinary commercial slags and is used where iron or other bases are in excess. It is still of too high specific gravity to give a perfect settlement of the matte and is seldom made where silicious ores are obtainable at a fair price. The **sesqui silicate** is a mixture of the singulo silicate with a bisilicate, and in the neighborhood of this silicate degree the majority of the copper slags may be placed. It has a low melting temperature and runs smoothly without forming a thick crust over its surface. It is low enough in specific gravity to permit a complete and rapid settling of the matte. It is perhaps the most desirable slag when conditions permit its use. The **bisilicates** are used in case of an excess of silicious ores. While they require a higher percentage of coke on the charge than the preceding slags, still when they are properly melted, give a slag of low specific gravity and a very clean separation of the matte. A **trisilicate** comes more within the range of the iron blast-furnace. The following table gives the limits of the different slag constituents as found in modern practice.

CONSTITUENT LIMITS FOR COPPER SLAGS AND FURNACE DATA

	Ordinary matte smelting.				Semi-pyrite smelting.				Pyrite smelting.				Reverberatory smelting.			
	Average, per cent.		Special, per cent.		Average, per cent.		Special, per cent.		Average, per cent.		Special, per cent.		Average, per cent.		Special, per cent.	
Constituents of slags.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.
SiO_2	44	28	52	25	47	..	52		Singulo silicate types				48	25	55	20
Al_2O_3	10	..	15	..	10	..	18	..	10	..	15	..	10	..	18	..
FeO	60	20	65	18	..	18	..	15	65	..	..	..	65	20	70	18
MnO	25	..	43	..	..	..	..	..	..	..	..	..	..	..	..	..
CaO + alkaline earths	28	10	40	6	30	10	40	..	28	10	..	8	28	10	40	0
MgO	5	0	12	0	5	0	15	0	5	0	15	0	5	0	18	0
$K_2O + Na_2O$	5	0	..	..	5	0	..	..	5	0	..	..	5	0	..	..
BaO	..	..	30	..	..	..	..	..	..	..	..	..	..	..	..	..
$BaSO_4$	..	..	5	..	..	..	..	..	..	..	..	..	..	..	..	..
ZnO	6	0	10	0	6	0	15	0	6	0	15	0	6	0	15	0
Cu	0.8(1.0)	0.5	0.5	0.2		same				same				same		
S	1.5	1.0	1.8	0.8		same				same				same		
Fe_2O_3	..	..	10	0	..	..	10	0	..	..	10	0	..	..	10	0
FURNACE DATA																
Zinc volatilized and going into matte	20	10	..	..	20	10	..	..	20	10	..	..	10	..	..	..
S volatilized and slagged	75	50	..	..	80	60	95	..	85	70	95	..	35	25	..	..
Fuel	15	10	17	..	15	3	..	..	3	1	..	..	35	25	bit. coal	
Copper on charge	15	..	..	..	..	0.5	..	..	..	..	..	..	..	..	..	..
Copper in matte	50	40	..	35	50	3	..	0.5	50	10	..	..	50	40	..	1
Copper plus iron in matte	65	..	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Sulphur in matte	25	23	29	20	..	..	..	..	..	..	..	..	..	..	..	..
Recovery of copper	95	90	..	..	..	same				same				same		
*Concentration ratio	7:1	..	10:1	..	10:1	..	22:1	..	10:1	..	22:1	..	7:1	..	..	..
Gold in slag	..	Trace.	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Silver in slag, oz.	1.0	0.25	..	0.20	..	same				same				same		

*By the concentration ratio is meant the ratio of the per cent copper on the 'charge' to the per cent in the matte.

By 'ordinary matte smelting' is meant that practice which uses raw or roasted ore with low concentration, of say 4 or 5 into 1, and 10 to 15% of fuel, and primarily for the reduction of copper ores with gold and silver as by-products. By 'semi-pyrite' smelting is meant that type which uses a high concentration ratio, silicious slags, as small amount of copper on the charge as possible, pyrite being used as a flux and fuel if available, but primarily smelting is for the collection of gold and silver from unroasted ores that are not suitable for amalgamation or leaching methods. By 'pyrite smelting' is meant a smelting of raw heavy sulphide ores for the collection of their copper content, also gold and silver, with the least possible addition of lime and silicious flux. It is not possible to sharply define all types of smelting by these three divisions as there will be gradations of one into the other as conditions require. Ores for regular pyrite smelting should contain 60% of pyrite or its equivalent.

Example.—The use of the table may be illustrated as follows. In deciding on a slag for smelting a copper ore by ordinary matte smelting it is desirable to see how high an alumina content may be used. Opposite Al_2O_3 in the column headed 'Constituents of Slags' and in the column headed 'Ordinary Matte Smelting,' 'Average % Max.' (maximum) is found the figure 10 which indicates that under average conditions 10% Al_2O_3 may be used in a slag. In a like manner under the column headed 'Special % Max.', is the figure 15 which indicates that in special cases

15% Al_2O_3 may be used in a slag. These figures are supplemented by the remarks on alumina which are given following the table. The table may be used in a like manner to find the average figures for other slag constituents for all common types of smelting, also other data as matte fall, fuel on charge, etc. The column heading 'Min.', is intended as an abbreviation for minimum. These columns contain the minimum or lowest figures found in average as well as special cases.

GENERAL REMARKS

Arsenic and Antimony.—These constituents are volatilized to a large extent in pyrite or semi-pyrite smelting. They may be almost entirely eliminated in reconcentrating the matte.

Alumina.—The following deductions may be made from Hoffman's investigations of alumina replacement:

Alumina cannot successfully replace SiO_2, hence with much Al_2O_3 present the SiO_2 must be kept high. Alumina cannot replace FeO to any extent with an undue rise in the temperature of formation of the slag. It is within reason to figure Al_2O_3 as replacing CaO. Taking all of the above into consideration shows that the formation temperature of the slag may be kept unchanged if alumina were figured as replacing part of the CaO and SiO_2. This method found practical application in smelting the aluminous Cripple Creek ores in silver-lead blast-furnaces. Although the temperature of formation of aluminous slags is low, yet they require much superheat to make them flow properly. In one case with slags between a sesqui silicate and a bisilicate, alumina was satisfactorily figured as a base. In another case with a higher percentage of alumina it was more satisfactorily figured as having two-thirds the replacement power of silica.

Baryta.—The same remarks apply as given under lead slags. The Tyee, British Columbia, plant ran slags which contained: SiO_2, 34.7%; FeO, 17.7; BaO, 28.5. A still higher percentage might be used in exceptional cases.

Heavy Spar ($BaSO_4$).—This constituent is partly reduced to BaS and enters the slag and matte, decreasing their difference in specific gravity, besides making thick infusible slags and tending to form wall accretions.

Galena.—The lead tends to volatilize and carry silver with it, and is an unwelcome addition to the charge on this account.

Iron and Manganese.—Their behavior is about the same, and MnO may be safely figured as FeO up to 28 to 30% MnO.

The following is an example of a slag having high SiO_2 and CaO and low FeO; reported as fluid, flowing readily but chilling quickly. It was used at the Rapid City, South Dakota, smelter.

	Per cent.
SiO_2	50.20
FeO	16.35
CaO	28.30
Al_2O_3	4.20
Total	99.05

When FeO alone is available for a base the following slag might be used:

	Blast-furnace. Per cent.	Reverberatory. Per cent.
SiO_2	38	38-42
FeO	62	62-58

Lime and Magnesia.—When using the maximum amounts of MgO, the slag is much less fluid than the temperature of formation would indicate and requires careful handling. It is essential while using high magnesia slags to keep up the percentage of iron to at least 20. With 15 to 18% FeO the issue will be doubtful. ZnO and MgO form a troublesome combination. The effect of replacing CaO with ZnO is to lower the temperature of formation, but the resultant slags show so little fluidity that much superheat is necessary for them to flow properly. When within reasonable limits, as given, MgO is a good flux as it has 1.4 times the fluxing power of CaO. The effect of high lime slags on formation temperatures is clearly shown in the table of slags following. The effect of lime is to lessen the specific gravity of the slags, which is productive of more perfect matte settlement, providing the slag has a sufficiently low temperature formation so that it may receive enough superheat to flow properly.

Sulphur.—The volatilization depends upon the nature of the ore. Under similar conditions a roasted ore will give a lower sulphur loss than an unroasted one. The conditions under which the furnace is run also governs the loss of sulphur; with high fuel, giving a strong reducing effect, less volatilization is to be expected than with low fuel and a large volume of blast.

Zinc.—The zinc entering the slag will depend upon the amount volatilized and going into the matte. The same remarks apply as given under zinc in lead slags. Under the roasting conditions for copper ores much of the ZnS is left undecomposed and enters the slag. Since zinc is volatile it causes a loss of silver by volatilization, forms wall accretions, condenses with the flue dust, and is thus returned to the furnace. When it is present in the charge to the extent of 1.5 to 2% the bad effects are noticeable, and 10% is prohibitive.

Mattes and Concentration Ratios.—It is not often desirable to use a matte of less than 5% copper, for efficienct collection of values. In cases where copper is lacking to bring the matte up to this standard, it would be advisable to re-charge enough roasted matte to reduce this, providing the additional saving paid for the cost of the operations. The copper in the matte should be proportional to the matte fall. With high concentration, 10% copper, with low concentration 2 to 3% copper is sufficient. The grade of the matte should also vary directly with the amount of gold and silver and impurities present. In general, it is not good practice to use high concentration ratios such as 18 to 22 tons of charge into 1 ton of matte, except where conditions demand this or with exceptionally pure ores in furnaces of 200 or more tons capacity. A 10 into 1 or 12 into 1 concentration is about the most satisfactory limit. For example in a furnace 36 by 84 inches, making a slag assaying SiO_2 34%, FeO 33, CaO 20, concentration of 10 into 1 caused the furnace to crust badly and often necessitated refeeding of matte. A 15% matte fall proved more satisfactory.

When the matte from the first smelting does not come up to converting or shipping grade, say 35% copper, it may be re-concentrated in the same furnace, or if the daily output is sufficient, in a separate furnace. The charge may consist of large lumps of raw matte, silicious ores, limestone, and rich slag. The process is inexpensive as computed per ton of original ore, and serves to remove As, Sb, Pb, and Zn which are detrimental to refining. The grade of matte produced for converting ranges ordinarily between 35 and 50% but more often between 45 and 50% copper.

Peters says, "I would advise no metallurgist to undertake a new enterprise (semi pyrite smelting) unless he sees his way clear to keeping the SiO_2 down toward 45% and his FeO up to at least 20%. If much Al_2O_3 is present, I would advise that in his preliminary calculations, he lower the SiO_2 in his slag 0.66% for each unit of Al_2O_3 in the mixture above 5%. If ZnS above 5% be present, I would allow an increase of 1% FeO in my slag for each unit excess ZnS in my mixture." He further adds that these figures are purely empirical, and gives other cautions.

The size of the blast-furnace has a decided effect upon the permissible limits of the slag constituents.

On account of the small amount of the slag and matte flowing away from the furnaces of low capacity and hearth area, a much more fusible and easy running slag must be used than would be necessary in furnaces of large capacity used in modern practice. Intermittent tapping will allow the running of slags that could not be handled with a continuous flow into a settler. For furnaces of less than 150 tons daily capacity, the following limits should be observed when possible: SiO_2 30 to 38%; FeO 30 to 40; CaO 10 to 25. One of the best all round slags has the following composition: SiO_2 36%; FeO 33; CaO 21.

Loss in Weight Due to Roasting.—When an ore is assumed to be roasted, as must often be done in preliminary calculations, it is necessary to approximate its analysis without resort to actual trial. Since oxygen (16) is one-half the atomic weight of sulphur (32), in roasting from sulphides to oxides the loss in weight can be approximated as one-half the loss of sulphur. Example:

Analysis original ore.	Per cent in 100 lb. ore.
SiO_2	60 or 60 lb.
Fe	20 " 20 "
S	20 " 20 "
Total	100 in 100 lb.

Roasted to a product containing 5% sulphur: loss in weight $= 100 - \frac{20-5}{2} = 92\frac{1}{2}$ lb.; therefore in $92\frac{1}{2}$ lb. of roasted ore there is SiO_2 60 lb., Fe 20 lb.

The new approximated analysis would be:

Per cent.

$SiO_2 \quad \frac{60}{92.5} \times 100 = 65.0$

$Fe \quad \frac{20}{92.5} \times 100 = 21.6$

S assumed $= 5.0$

The following slag tables (by Hoffman) show the variation of constituents possible in the different silicate degrees with the attendant effect on the temperature formation. The metallurgist must not be entirely guided by the low formation points of the highly silicious bisilicates as they need a temperature considerably higher to flow properly. They may be used successfully in semi pyrite smelting on account of the high temperature attained.

HOFFMAN'S SLAG TABLES

Subsilicate $3RO,SiO_2$.				3 to 4 Silicate $3RO,2SiO_2$.			
Chemical composition of slag.			Melting point.	Chemical composition of slag.			Melting point.
SiO_2, %	FeO, %	CaO, %	Deg. C.	SiO_2, %	FeO, %	CaO, %	Deg. C.
21.70	78.30	0	1220	35.70	64.30	0	1140
21.95	74.05	4	1230	36.05	60.00	4	1110
22.20	69.08	8	1220	36.40	55.60	8	1090
22.49	65.51	12	1200	36.80	51.20	12	1070
22.70	61.30	16	1240	37.30	46.70	16	1090
22.95	57.05	20	1250	37.75	42.25	20	1110
23.20	52.80	24	1210	38.16	37.84	24	1130
23.45	48.55	28	1190	38.56	33.44	28	1150
23.70	44.30	32	1170	38.95	29.04	32	1160
23.94	40.06	36	1170	39.37	24.63	36	1170
24.20	35.80	40	1230	39.78	20.22	40	1190
24.45	31.55	44	1310	40.20	15.80	44	1290
24.48	27.52	48	1430+	40.60	11.40	48	1430+
24.95	23.05	52	1430+	41.02	6.98	52	....

Singulosilicate $2RO,SiO_2$.

Chemical composition of slag.			Melting point.
SiO_2, %	FeO, %	CaO, %	Deg. C.
29.20	70.80	0	1270
29.75	66.25	4	1250
30.09	61.91	8	1240
30.42	57.58	12	1220
30.76	53.24	16	1170
31.07	48.90	20	1205
31.40	44.60	24	1190
31.70	40.30	28	1170
32.10	35.90	32	1150
32.30	31.70	36	1130
32.70	27.30	40	1150
33.10	22.90	44	1190
33.44	18.56	48	1270
33.79	14.21	52	1430+

Bisilicate RO,SiO_2.

45.45	54.55	0	1110
46.00	50.00	4	1170
46.53	45.47	8	1030
47.04	40.96	12	1050
47.56	36.44	16	1090
48.02	31.98	20	1130
48.57	27.43	24	1170
49.19	22.81	28	1200
49.60	18.40	32	1250
50.11	13.89	36	1330
50.63	9.37	40	1430
51.14	4.86	44	....
51.65	0.35	48	....
51.73	0.00	52	....

Sesquisilicate $4RO,3SiO_2$.

Chemical composition of slag.			Melting point.
SiO_2, %	FeO, %	CaO, %	Deg. C.
38.46	61.54	0	1120
38.90	57.10	4	1090
39.34	52.66	8	1060
39.78	48.22	12	1060
40.22	43.78	16	1090
40.66	39.34	20	1130
41.11	34.89	24	1150
41.54	30.86	28	1160
41.99	26.01	32	1165
42.42	21.58	36	1190
42.87	17.13	40	1250
43.31	12.69	44	1330+
43.75	8.26	48	....
44.19	3.81	52	....

Cross series, FeO:CaO = 2:1.

18.67	54.23	27.10	1190
25.61	49.60	24.79	1180
31.47	45.68	22.85	1190
36.47	42.36	21.17	1180
40.80	39.46	19.74	1160
44.55	36.97	18.48	1140
47.86	34.77	17.37	1120
50.82	32.78	16.40	1115
53.44	31.04	15.52	1110
55.81	29.46	14.73	1110
57.95	28.04	14.01	1130
59.87	26.75	13.38	1310+

CALCULATION OF A SILVER-LEAD BLAST-FURNACE CHARGE

Conditions.—Assume that it is desired to smelt as much dry silicious ore as is possible with only enough lead for a collector and that zinc sulphide is present in the lead ore to an extent which would lead to choice of a slag of the C type, which is especially adapted to working zincy ores and at the same time is as silicious as possible. It will be necessary to put enough roasted matte in the charge so that this product will not accumulate. Clippings, foul slag, etc., will have to be introduced after the plant is running regularly. The charge may be started with 50 lb. roasted matte, figuring the whole charge on the basis of 1000-lb. units for convenience. For the first, or starting charge the most favorable slag possible would be chosen so that the irregularities of starting may be more easily overcome. The analyses of ore, etc., are given on the accompanying charge sheet.

The next consideration is to get a safe amount of lead on the charge, assumed at 10%. This may be done by adding enough galena to furnish 100 lb. of lead or 167 lb. galena. Next, purely by guess and later by experience, 275 lb. silicious ore is put in as the probable amount required to make a charge of 1000 lb. The next step is to figure out the weights of the various constituents SiO_2, CaO, etc., The total weight of the SiO_2 from the galena, matte, and silicious ore, amounts to about 220 lb. This gives data enough to make the first estimate of the amounts of limestone and iron ore required.

Factors.—The Fe in the slag is $\frac{26.4}{34.0}$ or 0.78 times the silica. Knowing the pounds of silica going into the slag, the amount of iron required to go with it to make a slag of the given analysis, can be found by multiplying the weight of the silica in the charge by 0.78. $220 \times 0.78 = 172$ lb. Fe required or $172 \div 0.60 = 287$ lb. of iron ore since the latter only contains 60% iron. Since the matte will require iron and both the iron ore and limestone contain silica which is not included in the above amount, of 220 lb., a slightly larger quantity of iron ore, 325 lb. must be taken.* $220 \times 0.5 = 110$ lb. CaO or $120 \div 0.54 = 200$ lb. of limestone or, allowing for the extra silica, about 225 lb. would be required. The analysis of the coke ash must be converted into terms of coke. The coke has 10% ash. The ash contains 75% silica, then the coke will contain 10% of the 75 or 7.5 silica. Complete finding the weights of the constituents and add up the total amounts.

All the iron is not available for the slags as some of it goes into the matte. The weight of the matte must first be computed from the amount of sulphur available. The slag contains 0.8% sulphur. Since the silica is 34% of the slag the weight will be $267 \div 0.34 = 785$ lb. and will contain 785 times $0.008 = 6$ lb. of sulphur. Then allowing for 20% volatilization 20% of $23 = 5$ lb. of sulphur may be calculated, making a total loss of 11 lb. and leaving, $23 - 11 =$

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*The amount of iron in the incoming matte is 25 pounds.

12 lb. of sulphur available for the matte. This amount will make 12 ÷ 0.20 = 60 lb. matte containing 20% sulphur. To find the iron in the matte the copper must first be determined. Practically all the copper goes into the matte if sufficient sulphur has been provided. There is then 8 lb. of copper in 59.5 lb. of matte or (8 ÷ 60) × 100 = 13% copper. Then the per cent of iron is 60 — 12.9 = 47.1% Fe. The weight of the iron going into the matte will be 47% of 60 lb. or 28 lb. which must be subtracted from the total iron in the charge, leaving the iron available for the slag. If there were any arsenic present to form a speiss, 2.3 its weight must be subtracted from the total iron. The next step is to see if the iron and calcium oxides are present in the right amounts to form the slag given. (Computations C.) According to the factors the Fe is 0.78 times SiO_2. There is 267 lb. of SiO_2 present hence there should be 0.78 times 267 = 209 lb. of iron present, but only 206 lb. is available hence there is a deficiency of 3 lb. or in terms of iron ore, 3 ÷ 0.60 = 5 lb. The charge will have to be corrected by adding this amount and if the error is very large all the computations will have to be gone over. According to the factor the CaO is 0.5 times the SiO_2 hence 267 times 0.5 = 134 lb. CaO is needed. There is 139 lb. CaO on the charge which is an excess of 5 lb. CaO or 5 ÷ 0.50 = 10 lb. of limestone which is added to the charge.

If desirable the approximate amount of base bullion may next be calculated, by subtracting the weight of the lead in the matte and slag from the total weight in the charge. This, of course, does not account for other metallurgical losses which are offset in a measure by the impurities entering the bullion. Lead in the matte may be determined as follows: 60 lb. of matte assaying 15% lead would contain 60 times 0.15 = 9 lb. of lead. That in the slag may be found as follows: 785 lb. of slag assaying 0.8% would contain 785 times 0.008 = 6 lb. lead. Total 9 + 6 = 15 lb. of lead or 107 − 15 = 92 lb. base bullion.

Knowing the weight of the base bullion its probable assay may next be computed. First compute the total weight of the gold and silver in the ores (not including the matte) from the assay values (Computation E), subtract from these amounts the weights going into the slag and the result will be the amounts going into the base bullion, not taking into account the other metallurgical losses such as volatilization and flue dust. The reason the matte is not taken into account is that, while silver and gold

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CHARGE SHEET

Name of Ore		Weight		Pb		SiO₂		Fe and Mn		CaO and MgO		S		Cu	
	H_2O	Wet	Dry	%	Wt.	%	Wt.	%	Wt.	%	Wt.	%	Wt.	%	Wt.
Roasted matte ---			50	15	8	2	1	50	25	1	1	8	4	12	6
Galena ---------			165	60	99	16	26			10	16	10	17	1	2
Silicious ore ----			275			70	192	4	11						
Correction --			330												
Iron ore ---------			325			10	33	60	.195						
Correction --			216												
Limestone -------			225			3	7			54	121				
Coke -------------	(150)					5	8	1.8	3	0.9	1	1.5	2		
Totals -------			1036		107 15 92		267		234 28 206		139		23 11 12		8

are introduced into the charge by the roasted matte, practically the same amounts will be taken out by the matte that is formed. Where matte is fed irregularly this short cut cannot be taken.

SLAG	Per cent.
SiO_2	34.0
FeO	34.0 (Fe, 26.4%)
CaO	17.0
Pb	0.8
S	0.8
Ag	0.5 oz.

COKE (10% ash, 1.5% S)

Coke ash.	Per cent.
SiO_2	75
Fe	18
CaO	9

MATTE	Per cent.
S	20
Pb	15
Cu + Fe	60

COMPUTATION (A) FACTORS

$$\frac{26.4}{34.0} = 0.78 \text{ iron factor.}$$

$$\frac{17.0}{34.0} = 0.5 \text{ lime factor.}$$

COMPUTATION (B)

$$\frac{267}{0.34} \times 0.8 = 6 \text{ lb. S in slag.}$$

$$22.7 \times 0.20 = 5 \text{ “ “ volatilized.}$$

$$\text{Total } \overline{11} \text{ “ “ lost.}$$

$$\therefore 23 - 11 = 12 \text{ “ “ to matte.}$$

$$\frac{8}{60} \times 100 = 13\% \text{ Cu in matte.}$$

$$\therefore 60 - 13 = 47\% \text{ Fe in matte.}$$

$$\frac{120}{0.20} = 60 \text{ lb. of matte.}$$

$$60 \times (0.60 - 0.13) = 28 \text{ lb. Fe in matte.}$$

COMPUTATION (C)

267 × 0.78 = 209 lb. Fe needed.
206 “ “ in charge.
3 “ “ lacking.
3 ÷ 0.60 = 5 lb. iron ore to be added.
267 × 0.5 = 134 lb. CaO needed.
139 “ “ in charge.
5 “ “ excess.
5 ÷ 0.54 = 9 lb. limestone excess.

COMPUTATION (D)

$\frac{267}{0.34} \times 0.008 = 6$ lb. Pb in slag.
$107 - (6 + 9) = 92$ lb. wt. base bullion.
$\frac{60}{1036} \times 100 = 5.8\%$ matte fall.
$60 \times 0.15 = 9$ lb. Pb in matte.

COMPUTATION (E)

Assays galena = Ag, 50.0 oz.
$\frac{165 \times 50.0}{2000} = 4.125$ oz. Ag in galena.
$\frac{267 \times 0.5}{0.34 \times 2000} = 0.197$ “ “ “ slag.
3.928 “ “ “ base bullion.
Silicious ore = Au, 1.00 oz.
$\frac{275 \times 1.00}{2000} = 0.137$ oz. Au in silicious ore.
trace “ “ slag.
0.137 “ “ “ base bullion.

ASSAY OF BASE BULLION

$\frac{2000 \times 3.928}{92} = 81.0$ oz. Ag per ton.
$\frac{2000 \times 0.137}{92} = 2.98$ oz. per ton Au.

CALCULATIONS OF A COPPER BLAST-FURNACE CHARGE

The problem is to select the best charge to smelt an ore containing SiO_2, 50%; Fe, 15; CaO, 10; S, 20; Cu, 5; gold 0.05 oz.; silver, 10.0 oz. under the following conditions: (1) furnace capacity 175 to 200 tons per 24 hours; (2) limestone close at hand; (3) iron ore must be shipped in 100 miles by railroad and contains no gold or silver; (4) coke obtainable at reasonable prices and of good quality.

Judging from the above conditions it would seem advisable to volatilize enough sulphur to make a converting grade of matte, to use as high a silica slag as is allowable by the size of the furnace, perhaps a sesquisilicate, and substitute lime for the base as much as possible without making the slag too refractory. The table of slag formation temperatures shows that the lowest limit for iron is 21.6% without an undue rise in the temperature formation. The coke may be finally reduced on account of the sulphur available for fuel, but 14% should be used at first and reduced to the lowest possible limits with careful watching.

The methods of calculation are given in the following charge sheets and are performed in much the same manner as described in detail under the calculation of the lead charge. The data necessary for the computation are taken from the table and matter previously given.

CHARGE SHEET

Name of Ore	H_2O	Weight		Cu		SiO_2		Fe and Mn		CaO and MgO		S		
		Wet	Dry	%	Wt.	%	Wt.	%	Wt.	%	Wt.	%	Wt.	
Copper Ore----			475	5	24	50	238	15	71	10	48	20	95	
Limestone----			400			10	40			50	200			
Iron Ore -----			100			10	10	60	60					
Coke --------	(140)					7.5	10	1	1	1	1	1	1	
			*975		24 2 slagged 22		298		132 12 in matte 120		249		96 83 loss 13 to matte	1

*This may be increased to 1000 lb. by taking more copper, probably 485 pounds.

SLAG

	Per cent.
SiO_2	42.4
FeO	21.6 (Fe, 16.8%)
CaO	36.0
S	0.8
Cu	0.25

COKE (10% ash, 1.0% S)

Ash.	Per cent.
SiO_2	75.0
FeO	10.0
CaO	10.0

MATTE

	Per cent.
S	25
Cu + Fe	65
S (volatile)	80

FACTORS

$$\frac{16.8}{42.4} = 0.4 \text{ Fe factor.}$$

$$\frac{36.0}{42.4} = 0.85 \text{ CaO factor.}$$

SLAG LOSSES

$\frac{298}{0.424} = 704$ lb. slag, 0.25% of 704 = 2 lb. Cu slagged.
$704 \times 0.008 = 6$ lb. S in slag.
$96 \times 0.80 = 77$ " " volatilized.
83 " " total loss.

MATTE COMPUTATIONS

$13 \div 0.25 = 52$ lb. of matte.
$(22 \div 52) \times 100 = 42.3\%$ Cu in matte.
$298 \times 0.4 = 119$ lb. Fe needed.
120 " " present.
1 " " excess.

AVAILABLE IRON

$65 - 42.3\% = 22.7\%$ Fe in matte.
$52 \times 0.227 = 12$ lb. Fe in matte.
$298 \times 0.84 = 250$ lb. CaO needed.
249 " " present.
1 " " lacking.

These figures are near enough, but larger amounts should be corrected as done in the Lead Charge Sheet.

MATTE FALL

$\frac{52}{975} \times 100 = 5.3\%$ matte fall.

This is low and might well be increased at the expense of lowering the grade of matte.

Remarks.—If the corrections in the fluxes amount to much in the first estimate, say over 10 lb., the calculations should be gone over again. The charge may be increased to any quantity by multiplying all of the constituents by the necessary factor.

APPLICATION OF SLAG DATA

Since the basic problem is to select the best charge or slag to smelt a given ore or selection of ores, whether it is done for comparison of two methods or to determine the most economical slag, all resolve into the question, what mixture put into the furnace will yield the greatest net profit.

Example.—Given a collection of ores and fluxes whose analyses are given in the following tables:

	SiO_2, per cent	FeMn, per cent	CaOMgO, per cent	S, per cent	Pb, per cent	Cu, per cent	Au, oz.	Ag, oz.
Lead ore	16	..	10	10	60	2	..	50
Silicious flux or ore	70	4	..	4	..	..	1	..
Irony ore	10	60	..	..	..	..	..	..
Limestone	3	..	54	..	..	..	..	..
Coke ash, 10% coke.	75	18	9	..	..	..	..	..

The lead ore is the one to be primarily considered, it being the output of the mine in question. The charges against this ore delivered at the smelter bins is $2 per ton. The silicious ore or flux may be readily obtained as a custom ore and a treatment charge of $6 per ton imposed, returning 95% of the metals to the seller. Limestone costs 75c. per ton delivered from the company's quarry, and irony ore costs $4 delivered at the smelter. First decide if any of the ores should be roasted. The only ore with an appreciable amount of sulphur is the galena. The one very prominent argument in favor of not roasting is its high silver content. Since one unit of sulphur costs 25c. to handle, then in one ton of galena would be the following costs if not roasted:

10 units of sulphur at 25c. $2.50 per ton.

Charges if roasted.	Per ton.
Roasting H. & H.	$1.76
Losses Pb, 4% of 1200 lb., at 1c*	0.48
Losses Ag, 4% of 50 oz., at 50c	1.00
4 units of sulphur left, at 25c	1.00
Total	$4.24

*The 1c. equals net profit that would ultimately be obtained if one pound of metal were recovered as market lead.

Difference in favor of not roasting, $1.74 per ton. This does not take into account any benefits from the use of a sintered H. & H. product. From the charge-sheet may be taken the quantities of ore and flux making up one ton of blast-furnace mixture and figures of profit and loss determined as follows:

PER TON OF CHARGE*

Recovery, Pb 93%, Ag 95%, Au 98%.

CHARGES

0.21 tons limestone, at 75c	$0.16
0.32 tons irony ore, at $4	1.28
0.16 tons lead ore, at $2	0.32
Refining 0.046 tons base bullion, at $12†	0.55
Smelting 1 ton charge, at $3	3.00
Total charges	$5.31

*These figures are taken from the lead charge sheet.

†The cost of refining is assumed to cover the cost of placing all the metals in marketable shape.

RECOVERY

192 lb. Pb, at 4c‡	$7.68
7.6 oz. Ag, at 50c	3.80
Custom ore, 0.27 ton, at $6	1.62
Total recovery	$12.10
Charges	5.31
Profit	$6.79

‡Does not include freight to market.

Next the charge may be calculated with a different slag and estimates made of the profit with the more basic or acid type. Here inaccuracies creep in because, with the data available, it is not possible to determine exactly the decreased cost for smelting one ton of charge when using a faster running slag or to estimate without actual experiment the difference in the recovery of the metals. The cost computations are repeated with the new slag and the resulting net profit compared with the previous results and deductions drawn.

It will be noted that the gold content of the silicious ore is neither charged nor credited to the furnace charge. This will introduce a slight inaccuracy because this omission is based on the assumption that the recovery of the gold will only be the 95% which is returned to the ore seller. Since the recovery of gold is given as 98% there will be a difference of 3% in favor of the recovery, or about 16c. in this case, which is well within the limits of errors for the example.

COST DATA

GENERAL

When figuring costs and drawing up designs, it is to be understood the following figures are only approximate and are intended for use in making preliminary estimates only. The question of costs is a vital one. The success of a method of treating an ore depends wholly upon dollars and cents and not upon the fact that the process in question is a scientific possibility. In making final estimates the entire scheme must be carefully analyzed and itemized step by step, taking care to determine the cost of each item correctly. An underestimate of a single item may cause the financial failure of a project as well as ruin the engineer's professional reputation. Cost data may be correct for one mining camp and entirely wrong for another a few miles distant. After completing a cost sheet it is well to add from 5 to 10% to the total as a factor of safety.

ORE TREATMENT

AMALGAMATION AND CONCENTRATION

STAMPS, PLATES, AND VANNERS

	Per ton.	Stamp units.
California	$0.40—$0.50	10—40
Black Hills	0.40— 0.50	200
Alaska	0.28— 0.30	200
Gilpin county	0.75— 1.00	10—40

PAN AMALGAMATION

Cost per ton of ore treated equals	$2.35

COARSE CONCENTRATION, JIGS, ROLLS, AND TABLES

	Per ton.	Ton units.
Missouri	$0.25—$0.30	100— 400
Colorado and Utah	0.75— 1.00	100— 150
Large Montana plants	0.35— 0.50	1000—2000
Steam stamps, Lake Superior.	0.25— 0.35	1000—2000

COMBINATION MILLS

	Per ton.	Stamp units.
Wet stamping, vanners, and pans.	$1.50—$5.00	60—100

*COST OF CONCENTRATION MILLS

	Tons an. cap.	Cost per ton an. cap.
Coarse concentration of galena, (Joplin)	50,000	$0.12—$0.16
Mixed sulphides with fine grinding (Joplin)	15,000	0.70— 0.90
Frame structure, Coeur d'Alene, date of erecting, 1900	75,000	0.60
St. Louis S. & R. R. Co., steel structure, 1900 (Missouri)	300,000	0.80
Silver Lake mill, fine grinding, mixed sulphides, hillside site (San Juan, Col.)	75,000	1.33
Boston Con., fine grinding, copper ore, electric power furnished (Utah)	1,000,000	1.50
Garfield mill, Utah Copper Co., fine grinding, erected 1907 (Utah)	2,200,000	1.85
Ohio Copper Co., finished price, estimated (Utah)	1,000,000	1.50

*Ingalls.

CHLORIDIZING MILLS

	Per ton.	Stamp units.
Dry stamping, roasting, and pans.	$6.00—$7.00	60—150

MAGNETIC SEPARATION

Per ton	$0.25—$0.75

COST OF MAGNETIC SEPARATION PLANTS

	Tons ann. cap.	Cost per ton ann. cap.
New Jersey Zinc Co. (N. J.)	300,000	$1.75
Fine grinding, Wilfley tables, Wetherill separators	15,000	3.00— 4.00

OIL FLOTATION

The cost will be approximately the same as for wet fine-concentration under the same conditions and will not exceed the latter by more than 10 to 15c. per ton, with a smaller cost of installation and will give a higher extraction on certain ores that are difficult or impossible to treat by wet methods.

CHLORINATION

	Per ton.	Ton units.
Barrel process	$3.50—$5.00	150—200

GENERAL

ROASTING

	Per ton.
Heap and stall	$0.50—$0.75
Hand reverberatory	1.80— 2.25
Mechanical	0.35— 1.00
Chloridizing	2.00— 2.50

HORSE-POWER PER TON OF ORE TREATED PER DAY

Type of mill.	Mesh.	Hp.
Stamps and vanners	20—40	0.75—1.0
Coarse concentration	10—20	0.5 —0.8
Combination stamp	16—30	1.5 —1.75
Chloridizing stamp (dry)	to 16	2.0 —2.5
Chloridizing stamp (wet)	to 40	4.0 —4.5
Magnetic separator		0.25—0.5
Cyanidation (dry) roll-crushing	20—30	0.5 —0.8
Cyanidation (wet), stamp crushing and sliming	to 80	0.75—1.5

COST OF POWER PER HORSE-POWER DAY

Steam-power, non-condensing engines, coal at $4.50 = 16 — 18 cents.

Condensing engines = 14 — 16 cents.

Compound condensing engines = 11 — 14 cents.

For each $1 increase in cost of coal add 1½, 1¼, and 1c. respectively.

Electric-power, $40 — $65 per yearly horse-power = 11 — 18 cents.

MISCELLANEOUS ITEMS

	Cents.	
Stamp batteries:		
Wear and tear	7 —12	per ton ore
Labor	8 —16	" " "
Power	8 —12	" " "
Coarse crushing (Blake and gyratory)	2½— 3	" " "
Rolls, dry crushing	7 —10	" " "
Rolls, wet crushing	3 — 7	" " "
Huntington mills	7 —25	" " "
Tube-mills	35 —60	" " "
Filter pressing (Moore and Butters)	10 —30	" " "

CYANIDATION

	Per ton.	Ton units.
Dry-crushing with rolls and leaching	$1.50—$2.00	100—200
Wet-crushing, leaching, and filter-pressing, or wet-crushing, sliming, and filter-pressing	0.85— 2.00	100—200

CYANIDE PRACTICE, ITEMIZED COSTS

Cyanide, 18 to 20c. per pound.

Lime, ½c. per pound.

Zinc precipitation = 4c. per ton solution.

Labor, cost for actual treatment.

Monthly capacity, in tons.	Cost, cents,
1,000	37.4
1,500	25.8
2,000	18.7
2,500	15.0
5,000	7.4
7,500	6.0
10,000	4.6
15,000	3.5
20,000	2.6

	Per month.
Manager	\$100—\$200
Chemist	100— 125
Foreman	80— 100

Maintenance and repairs should be figured at from 8 to 10% per annum, calculated on the original cost of the plant. Clean-up and realization of the gold will cost 8 to 10c. per oz. of fine gold. Sundries will amount to 1 to 1½% of total cost of treatment. Depreciation may be figured at 10% per annum, of cost of plant; this is then reduced to figures based on tonnage by dividing by total number of tons treated per year. General charges may include a portion or all of the following expenses, which are beyond the control of the metallurgist: (1) general manager's salary; (2) consulting engineer's salary; (3) mine office expenses; (4) head or other office expenses; (5) directors' fees, etc.

A modern plant of steel construction, fully equipped for classifying and treating battery pulp, can be erected for \$300 per ton treated daily. Figures for plants of other design and construction may vary from \$180 to \$600 per ton treated daily. The itemized costs include: (1) excavations; (2) foundations and supports; (3) vat precipitation plant; (4) machinery; (5) buildings; (6) tailing disposal.

COST OF HYPOSULPHITE LIXIVIATION

DRY STAMPING AND CHLORIDIZING ROASTING

Crushing	$1.36
Roasting and salt (6%)	2.68
Labor in leaching	0.27
Chemicals	0.30
Superintendence	1.02
Heating, lighting, pumping, and repairs	0.75
Total cost per ton	$5.71

COST OF HUNTINGTON-HEBERLEIN POT-ROASTING

ASSAY OF ORE

	Per cent.
Pb	50
Fe	15
S	22
SiO_2	8
Al_2O_3, etc.	5

Flux used, 344 lb. of limestone and 130 lb. of quartz. The ore was 80% of the charge. It was planned to make a slag consisting of SiO_2 30%, FeO 40, CaO 20.

Items.	Cost.
Crushing 1 ton at 10c	$0.10
Mixing 1 ton at 10c	0.10
Roasting 1 ton at 63c	0.63
Delivering 1.1 ton to converter at 12c	0.13
Converting 1.1 ton at 60c	0.66
Breaking 0.9 ton at 60c	0.54
Total	$2.16

Cost per ton of ore is $2.16 ÷ 0.80 = $2.70.

Treatment by the Salvesberg process will cost less by the amount of the preliminary roast, which is 63c. per ton of charge or 79c. per ton of ore, making the costs respectively $1.53 and $1.91.

COST OF LEAD SMELTING

AVERAGE PER TON OF CHARGE AT DENVER AND PUEBLO, COLORADO

Coke	$0.84	
Powder, supplies, labor	1.66	
General expense	0.16	
Total	$2.66	exclusive of roasting

COST OF SILVER-LEAD SMELTERS

	Tons ann. cap.	Cost per ton ann cap.
Modern blast-furnace works; lower figure usual	330,000	$2.30— 3.00

REFINING LEAD BULLION

Parkes Process.—Cost = $3 to $5 per ton based on actual working costs to which must be added interest, expressage, brokerage and treatment of by-products, making the total approximately $10 per ton. The following is an itemized account:

Items.	Cost.	
Labor	$1.968	
Spelter	0.861	
Coal	0.496	
Coke	0.521	
Supplies, repairs, and general expenses	0.289	$4.135
Interest	1.317	
Expressage	1.085	
Parting and brokerage	2.121	
Re-working by-products	1.492	6.015
Total		$10.150

COST OF PARKES PROCESS REFINERY

	Tons base bullion.	
Modern equipment	30,000	$6.66

COST OF COPPER SMELTING

TENNESSEE COPPER CO., 1903

	Per ton.
Coke, 0.1283 tons, at $4.93	$0.6082
Quartz, 0.0958 tons, at 90c	0.0862
Supplies, including coal for power	0.1266
Labor and superintendence	0.2111
Total	$1.0321

LEADVILLE, COLORADO, 1900

Average cost for one plant, $3.64 per ton matte shipped.

MT. LYELL, TASMANIA, 1902

	Per ton.
Mining	$0.5002
Stripping	0.5000
Smelting	3.3648
Converting	0.3612
Total	$4.7262

This is the average cost for 159,450 tons ore, average assay Cu 2.36%, Au 0.069, Ag 2.23 oz.; 18,537 tons fluxes, average assay Cu 1.70%, Au 0.026, Ag 0.24 oz.; 185,689 tons custom ore; total, 183,676 tons.

MT. LYELL, 1905

	Per ton.
Labor, coke fluxes, re-treating flue dust	$0.70
Total cost, including all charges	1.50

This is the average cost for 159,450 tons ore, average ing 38 to 45% copper, from ore averaging 2.3% copper.

COST OF COPPER SMELTING WORKS

	Tons ann.	Cost per ton ann.
Blast furnace plant, semi-pyritic process, no roasting, 1901......	330,000	$1.70
Balaklala works, McDougall roasters, blast-furnaces, reverberatory furnace; cost includes 25c. for converter plant, 1907-08 (Cal.).	437,500	2.25
Washoe plant, cost includes concentration and discarded equipment (Mont.)	3,000,000	3.58
Highland Boy; total cost to date (Utah)	300,000	3.24
Garfield works; extensive provisions for expansion (Utah)...	800,000	7.50

COST OF COPPER LEACHING

Henderson Process.—Including crushing roasted pyrite clinker, muffle chloridizing roasting, leaching and precipitation, with common labor at $1.50 per day, $1.87 per ton.

COST OF ELECTROLYTIC COPPER REFINING

Items.	Cost.
Labor	$0.75
Current	1.50
Supplies, acid, coal, etc............................	0.67
Lighting	0.80
Working costs	$3.00
Interest on permanent capital invested (5% of $2,000,000)	3.20
Depreciation (10% of $507,000)......................	1.60
Total	$7.80

Based on a daily treatment of 85 tons of anodes.

BETTS ELECTROLYTIC PROCESS

Items.		Cost.
Power 7.6 hp. days total at $50 per electric horsepower year		$1.06
Tank room, platform, and repair labor		0.86
Melting lead, labor, supplies, and repairs		0.38
Coal for melting lead		0.13
Chemicals, 6 lb. SiF_6 at 6c	$0.36	
¾ lb. glue	0.07	
		0.43
Slime treatment, except power and assaying, including parting		0.96
		$3.82
Credit about 20 lb. copper at 3c		0.60
Net cost		$3.22
Other expenses same as in Parkes process		6.00
Total cost		$9.22

COST OF ZINC SMELTING

Items.	Cost.
Labor (not including repairs and renewals)	$6.62
Fuel, 3 tons at 75c. per ton	2.25
Fine coal or coke for reduction, ½ ton per ton of ore at 84c. per ton	0.47
Clay for retorts, 0.1 ton of ore at $2.60 per ton	0.26
Repairs, renewals, and sundry supplies, also putting repaired furnaces in operation (heating up)	0.75
Total cost per ton blende concentrate	$10.35

This includes cost of roasting in a hand-stirred reverberatory roaster.

COST OF ZINC SMELTERS

	Tons ann.	Cost per ton ann.
Natural-gas fuel with roasters (Kansas)	25,000	$8.00—10.00
Coal fuel, gas-producers, regenerative furnaces		18.00
Sulphuric acid works, additional cost		5.00— 6.00

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